MINING ENGINEERING

NOVEMBER 1950

In Two Sections—Section 1

New model K

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The new WILFLEY Model K Centrifugal Sand Pump features important mechanical improvements which give you substantial dollar savings—in power and production. This new pump delivers continuous, trouble-free operation—stepped-up production—dependable performance in handling sands, slimes or slurries. Individual engineering on every application—an economical size for every requirement. Write or wire for details.

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MINING ENGINEERING

Incorporating Mining and Metallurgy, Mining Technology and Coal Technology

VOL. 187 NO. 11 NOVEMBER 1950

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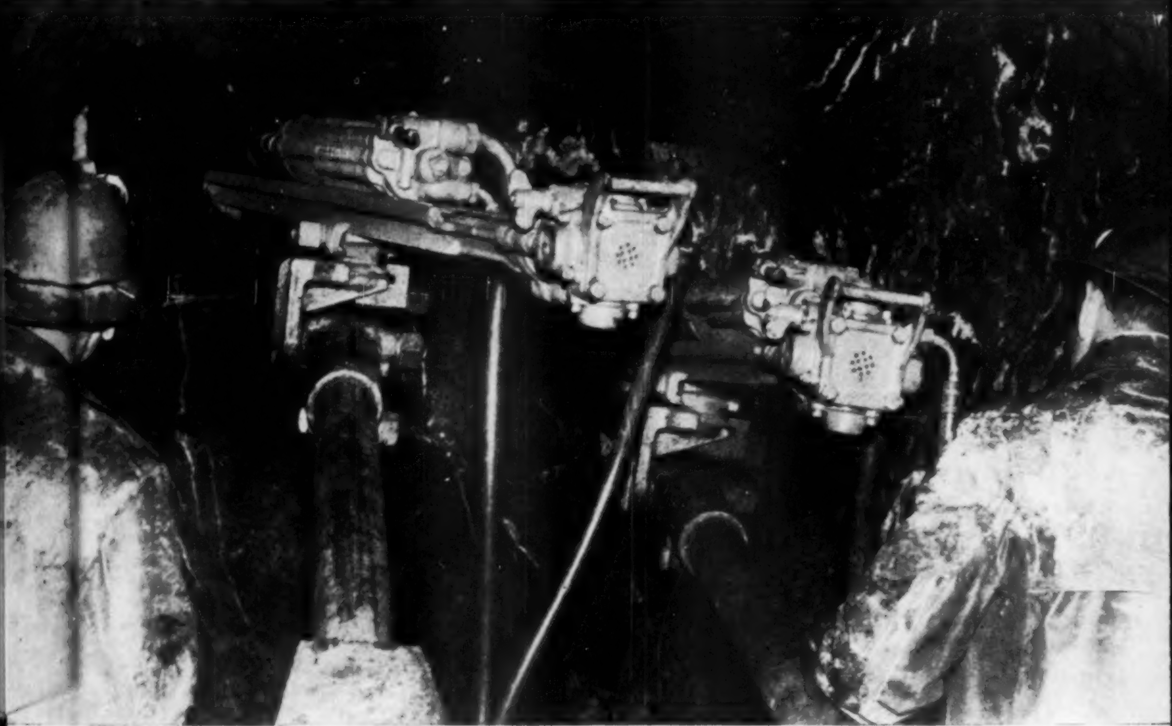
FEATURES

ARTICLES

TRANSACTIONS

COVER: Seventy-eight inch duplex high-wear Akins classifier operating in closed circuit with 9 x 8-ft Marcy ball mill in grinding and concentrating plant of Climax Molybdenum Co. at Climax, Colo.

Only JOY



Below, a JOY T-300 Drifter on column mounting in a small heading.

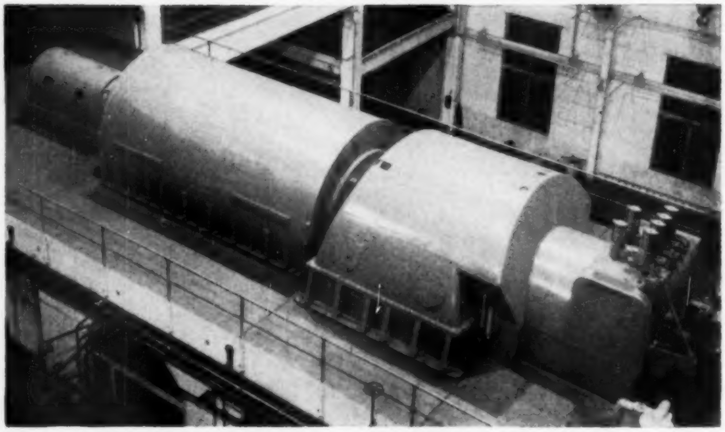
Above, two JOY T-350 Drifters mounted on JOY Hydro Drill Jibs.

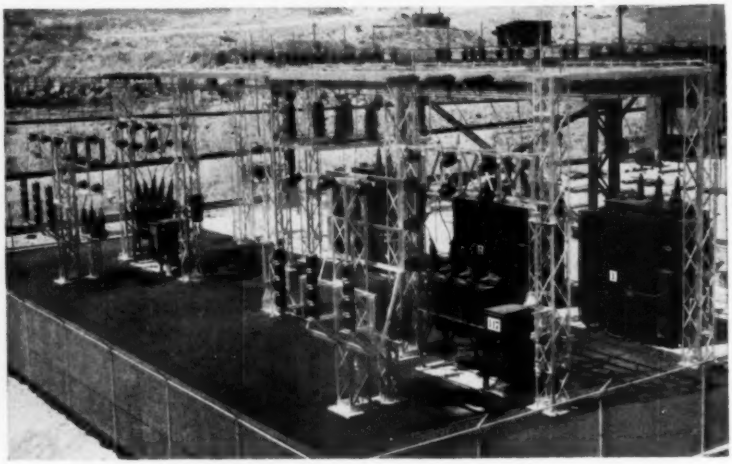
There's a Complete Line of JOY SILVER STREAK STOPERS with DUAL VALVE, CADMIUM PLATING, and the safe THUMB-FLIP rotation control.

Use JOY SULMET BITS—tungsten carbide tipped for longer life, and for more footage with fewer bit changes.

Powered for low-cost high-tonnage refining!

1 Typical of the mining industry's increasing electrification to handle high tonnages at low cost is this new copper refinery at Garfield, Utah, which has a capacity of 12,000 tons of refined copper per month, with provision for expansion to 16,000 tons. Power for the refinery and other operations comes from Kennecott's power station, where three G-E steam turbine-generators (two of 25,000 kw each, plus the 50,000-kw unit shown here) generate 13.8-kv power which is stepped up to 44 kv for transmission.





2 At the refinery, three miles away from the power plant, incoming 44-kv power is stepped down to 13.8 kv at this G-E package substation. Completely co-ordinated, it includes an outdoor steel switching structure, three 7500/9375-kva power transformers (with provision for addition of a fourth) and necessary metal-clad switchgear. G-E package substations, made in many standard combinations to fit all mining-industry needs, come in factory-built sections ready to install. They simplify ordering, save engineering time, speed installation, cut costs.

GENERAL ELECTRIC

660-20

Authors in This Issue


R. T. Hukki


J. Griffen


F. S. McNicholas


H. H. Fields

F. S. McNicholas (P. 1116), consulting engineer for the Climax Molybdenum Co. and for the Mufulira Copper Mining Co., was for ten years assistant general superintendent at Climax. He has also spent 10 years as assistant mine superintendent for Granby Consolidated in Anyox, B. C. In addition he has been with Anaconda at Butte and Condo, with the Buchana Mining Co. in Newfoundland, and with the Chicago Mining Co. in Colombia, S. A. Mr. McNicholas graduated from the Colorado School of Mines in 1914. He has written or co-authored nine articles and technical papers for *AIME Transactions, Mining and Metallurgy,* and *Engineering and Mining Journal.*

F. M. Lewis (P. 1133), assistant superintendent of mills at Tennessee Copper Co., Copperhill, Tenn., has been with the company since his graduation from the University of Tennessee in 1926. He lives in Copperhill, is an AIME member.

J. Griffen (P. 1137), consulting engineer for the McNally-Pittsburgh Mfg. Co., working on design and sales of bituminous coal preparation plants. He has a Ch.E. degree from Lehigh University, and has worked on various coal preparation and cleaning problems for Lehigh Coal and Navigation, The Hudson Coal Co., The Dorr Co., and American Rheolaveur Co. He's an AIME member and has authored numerous papers for the Institute. Enjoys gardening, stamp collecting, and golf.

D. M. McFarland (P. 1123) is a graduate of MIT with an S.B. in chemical engineering. He began his career with the Atlas Powder Co. at their experimental laboratory in Tamaqua, Pa., in 1920, joined the Company's technical division 7 years later, and has been manager of that division since 1934. His headquarters are in Wilmington, and he lives in West Chester, Pa. In Dec. 1948 we published his "Better Fragmentation Claimed for Fast-Delay Caps" in *Mining and Metallurgy.* His leisure time is devoted to gardening and hunting.

R. T. Hukki (P. 1131) is a native of Finland, and he took his degree in mining engineering at the Finland Institute of Technology. He also has a B.Sc. from Queen's University, Kingston, Ont., Canada, and an Sc.D. from MIT. He has done research work at MIT, spent a year with Phelps-Dodge at Morenci, Ariz., and is now professor of mineral dressing at the Finland Institute of Technology. Professor Hukki is an AIME member, and co-authored "Principles of Comminution" with A. M. Gaudin in Transactions for 1946.

U. Runolinna (P. 1131) was born in Finland, attended high school in Helsinki, and took his degree in mining engineering from the Finland Institute of Technology. He is now a research engineer at the State Research Institute in Helsinki.

H. H. Fields (P. 1110) has his B.S. from the University of Minnesota and an E.M. from the Michigan College of Mines. He has been an engineer for Cerro de Pasco in Peru, worked for Utah Consolidated Mining Co. at Bingham, and has worked in Alaska as an examining engineer. His leasing activities in Utah and Colorado were followed by a period as an ore buyer for AS&R in Denver and El Paso, and more recently he has been an independent operator in Arizona and Mexico. He lives in Carbo, Sonora, Mexico. Of his vanished leisure he says: "A small mine operator has no time for hobbies. When I was an engineer for AS&R I found time to play golf, but not now."

R. C. Ferguson (P. 1127) was born in Yale, Mich., and attended the Michigan College of Mining and Technology, taking his B.S. in 1925. Since graduation he has been with the Hardinge Co., serving one year as draftsman, 13 years as assistant chief engineer, 2 years as field service engineer, and 2 years as head of the government liaison department before assuming his present post.

Engineering Societies Personnel Service

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18**; 100 Farnsworth Ave., **Detroit**; 57 Post St., **San Francisco**; 84 E. Randolph St., **Chicago 1**. Applicants should address all mail to the proper key numbers in care of the New York office, and include 6¢ in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter or $12 a year.

MEN AVAILABLE

Production Superintendent or Manager, graduate mining engineer, experience in metallurgy and mining, nine years in nonmetallics processing, including calcining. Married, two children, 35 years of age. Experience and ability in production, plant engineering and maintenance, plant processes, packaging, material handling, warehousing and shipping, labor supervision. Accustomed to responsibility. Best references. M-593.

Mining Engineer and Metallurgist with management experience, desires position as manager or superintendent of small, sound property. Twenty-five years in underground work, engineering and flotation plants treating complex base metal ores. Design and construction. Age 44, married. Cost conscious, capable. Employed; available thirty days. M-594.

Maintenance Engineer, 32, married, 10 years' experience in drafting, grade work and civil engineering, desires responsible position in construction or machinery erection field. Can report for interview. Employed. M-595.

POSITIONS OPEN

Mill Superintendent, 35-45, with at least 10 years ore milling experience, to supervise operations at lead-copper project. Salary, $5200-$7000 a year. Location, Midwest. Y4295-C.

Geologist for staff position in research laboratory, to act as geologist, sedimentologist in connection with oil field secondary recovery project. Location, western Pennsylvania. Y4244.

Assayer-Chemist familiar with the methods of wet analysis, as well as fire assay. Must be single. Salary, $2700 a year plus room and board and transportation. Location, Central America. Y4194.

MINING ENGINEER AND GEOLOGIST—45. Available for temporary assignment up to six months. Experienced in examination, exploration, and development metal mines and prospects. Qualified for domestic or foreign work.

Box J-19—MINING ENGINEERING

WANTED. . experienced master mechanic by a company operating several iron ore mines. Applicant must have substantial amount of experience in maintaining shovels (both electric and diesel), diesel trucks of the heavy duty type, conveyor belt installations, and general maintenance work around mines and concentrating plants.

Box J-21—MINING ENGINEERING

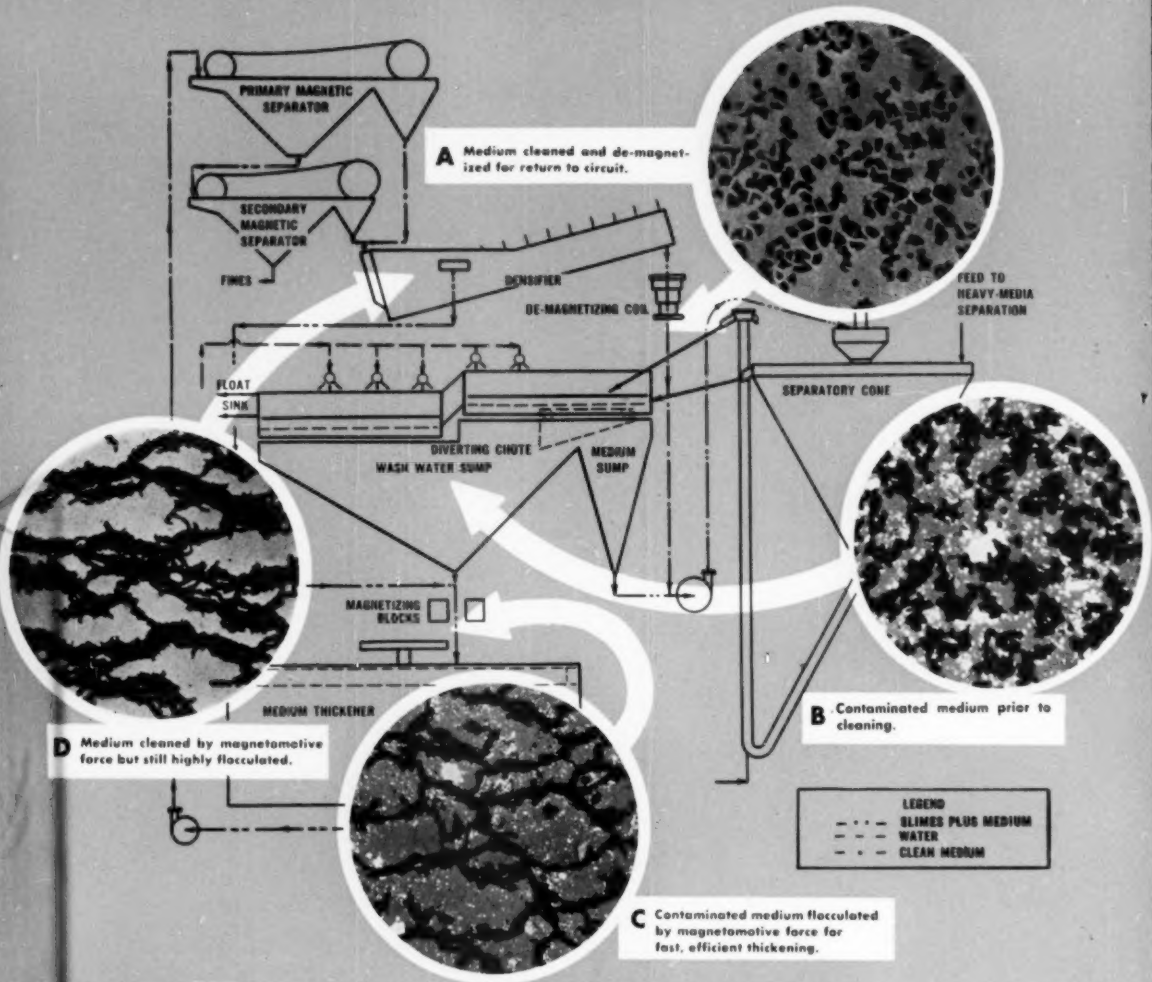
CHEMIST-ASSAYER, college graduate, competent, experienced in fire assaying and wet determinations of lead, zinc, copper, gold, silver, tin, tungsten, bismuth, antimony, etc. ores and concentrates, in charge laboratory employing twenty workmen and handling about 10,000 assays monthly, standard three-year contract, working knowledge Spanish essential, base salary $4200 yearly, plus yearly bonus one month, single status or if married single status for six months, free transportation to Bolivia by air for employee and wife, free living quarters, four weeks' vacation yearly.

Box J-18—MINING ENGINEERING

Large mining company in South America presently designing metallurgical plant involving new electric smelting process desires employ at earliest date possible talented metallurgist with proven operating experience, preferably between 28 and 34 years old. Location New York for approximately one year during design phase, engaging in technological studies, observing practices in other plants using somewhat similar techniques, then to move to property during middle and end of construction period, with good prospect of ultimately supervising plant. Three-year contract, living allowance while on temporary duty in U. S. Unusually attractive opening for able and ambitious man wishing to enter new field offering great possibilities.

Also require services two younger men with metallurgical training to work closely with above man, eventually to become operating foremen.

Reply Box J-20—MINING ENGINEERING



Photomicrographs show how

Heavy-Media Separation provides the most accurate separation by specific-gravity difference in the range 1.25 to 3.40

For years, mineral-dressing engineers have recognized heavy-liquid tests as the standard of comparison for mechanical or gravity separation processes in the beneficiation of metallic or non-metallic minerals. For years, they have known that a float-sink process using a mixture of finely ground, heavy solid and water offered the greatest promise of equalling on a commercial scale the perfect gravity separations obtained in heavy liquid laboratory tests.

Certain magnetic solids proved to have the ideal characteristics: High specific gravity, abrasion and corrosion resistance, ready availability and economic cost. With ferrosilicon and water or magnetite and water, accurate separations could be made over the wide gravity range of 1.25 to 3.40.

But despite these inherent advantages, the use of float-sink methods employing fine magnetic solids never achieved commercial success until the development of Heavy-Media Separation Processes. The signal success of Heavy-Media Separation is based on unique and exclusive applications of magnetomotive force to provide the necessary constant control, conditioning and recovery of the medium.

Photomicrographs of medium samples taken from a typical operating plant at the indicated points in the flow scheme show clearly how Heavy-Media Separation provides this precise and automatic medium control.

A shows the clean, dispersed and uniform medium that must be continually returned to the separating pool in order to maintain the desired specific gravity and to keep the viscosity low enough so that it will not interfere with sharp separation. By controlling the amount of medium sent through the cleaning circuit, the operator can maintain the specific gravity in the separating pool within ± 0.01 of the optimum separating gravity.

B shows dirty medium washed from the concentrate and gangue. Black areas are particles of ferrosilicon. White and grey spots are ore slimes that must be quickly and constantly removed so that they will not lower the gravity or raise the viscosity of the separating medium.

C shows the dirty medium after passing through magnetizing blocks. It is now well flocculated for fast settling in the medium thickener where the ferrosilicon quickly sinks while slimes and excess water overflow. Magnetomotive flocculation permits the use of relatively small, efficient thickeners with lower medium losses as well as lower plant costs.

D shows the medium after passing through the 99.9+% efficient magnetic separators. Non-magnetic foreign matter has been eliminated. The medium is now clean, but still in a flocculated, fibrous, magnetized state. To destroy the magnetic charge and allow the medium to disperse readily, it next passes through a demagnetizing coil. Now clean, dispersed and comparable in quality to its original condition (shown in A), the medium is ready to return to the separating pool.

No other float-sink method provides this exclusive 4-stage medium control with attendant low medium cost. No other method—float-sink or mechanical—provides the same uniform excellence of separating efficiency hour after hour, regardless of normal fluctuations in rate or character of feed.

More than 68 Heavy-Media Separation plants are now in operation for the beneficiation of 16 different minerals: Zinc, lead-zinc, iron, tin, manganese, lead, fluorspar and chromite ores, anthracite and bituminous coal, gravel, diamond-bearing ground, spodumene, andalusite, magnesite and garnet. As a pre-concentration method and to make a finished concentrate, Heavy-Media Separation has proved more accurate and economical than any other mechanical separation process on sizes down to 30 mesh. Indications are that feeds as fine as 65 mesh may be treated profitably in some instances.

Before you decide upon any mechanical or gravity separation method, it will pay to make a comparative study of the higher recovery and better grade you can obtain by using Heavy-Media Separation. American Cyanamid Company, as Technical and Sales Representative for Heavy-Media Separation Processes, will work closely with you in the design and initial operation of Heavy-Media Units. We are prepared to run tests in the Heavy-Media Laboratory and to give you every practical help in fitting Heavy-Media Separation into your flow scheme.

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MINERAL DRESSING DIVISION

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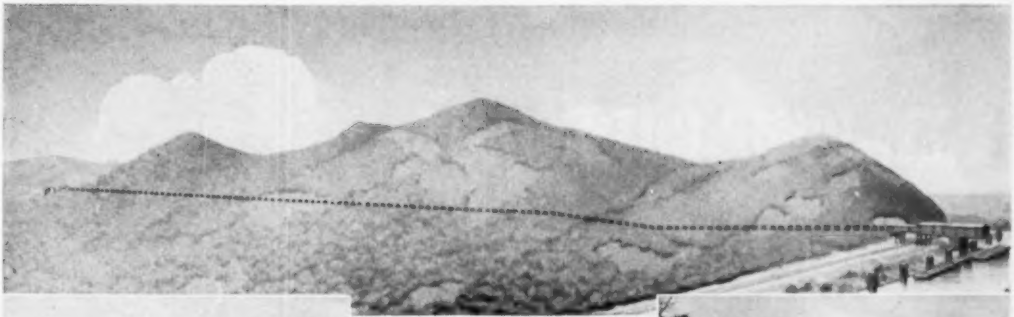
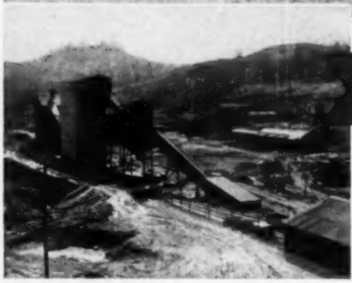




Here coal is prepared in this new blending and washing plant completely designed and built by Link-Belt. From this plant the washed coal is delivered to the tunnel belt conveyor (right side of above photograph) on which it is transported to the river and rail loading station.

TWO MILES IN ONE FLIGHT!

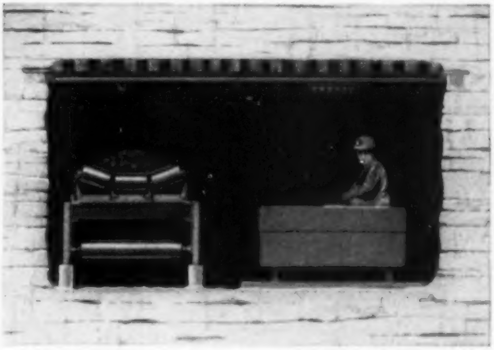
River and rail loading station also built by Link-Belt Company where washed coal is loaded into barges, or by means of a by-pass conveyor, into railroad cars.

World's Longest Single Belt Conveyor

Through a mountain, under forests, roads and streams this single 30" wide belt conveyor transports coal from preparation plant to river and rail loading station—10,900 ft. from foot pulley to head pulley—more than four miles of belt operated by one drive!

After careful analysis of various methods of transportation ultimate economy dictated the selection of this belt conveyor. By building this conveyor in one flight, intermediate transfers, heavy machinery and power wiring were eliminated from the tunnel.

Link-Belt Company engineered, equipped and erected the blending and washing plants, the conveyor equipment and the river and rail loading station. Resulting success of this and other similar projects illustrates the importance of such coordinated effort.



Typical cross-section through tunnel showing belt conveyor and battery driven patrol car. Standard Link-Belt type "100" idlers are used throughout the 2-mile long belt conveyor.

LINK-BELT COMPANY

Chicago 9, Indianapolis 6, Philadelphia 40, Atlanta, Houston 1, Minneapolis 5, San Francisco 24, Los Angeles 33, Seattle 4, Toronto 8, Johannesburg. Offices in Principal Cities. 11,944-A

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Belt Conveyor

EQUIPMENT

IDLERS • TRIPPERS • BELTS

PULLEYS • BEARINGS • DRIVES

* A loan, perhaps as great as $100 million, from American insurance companies is about negotiated for development of Labrador iron ore properties. The program will call for a total expenditure of up to $200 million, including, as well as all mine development work, the power facilities, housing for mining personnel, and auxiliary equipment.

* Mobile air compressors and a water tank, mounted on a 30-ton rubber-tired trailer and towed right up to the drilling crews, eliminates the need for expensive underground water and air pipe lines in the experimental oil shale mine operated by the Bureau of Mines near Rifle, Colo.

* Acceptance of roof bolting is evidenced by the fact that more than 350 major coal producers are using roof bolts to hold up about 37 million sq ft of roof surface. Information Circular 7583 of the U. S. Bureau of Mines describes safe practice in this method of roof support.

* Although only a little larger than the state of Utah, Korea is rich in mineral resources. In 1948, 3.6 million lb of tungsten ore were shipped to the U. S. It also produces graphite, fluorspar, and molybdenum in important quantities. In addition Korea has iron, aluminum, cobalt, copper, nickel, magnesite, lead, vanadium, and zinc mines.

* Army Engineers saved $4,349,000 in one year by eliminating "plugging" of dredge pumps. A gas ejection system connected to pump suction cuts pumping time 50 pct, permits dredging usually considered uneconomical.

* Clear salt solution as a street surfacer is now being used for the first time on a large scale in New Orleans to form a dustless, hard surface on shelled streets. Calcium chloride, bought in powder form and mixed with water to form liquid salt is also effecting economy in the maintenance of city streets.

* Zinc, lead, and other metals are being recovered commercially from slag piles at three plants now in operation. Designed by Babcock & Wilcox Co., these plants recover the metals by slag fuming or remelting the slags and collecting the fumes. These vapors are converted into metallic oxides, and then collected as solids or powders.

* British Aluminum Co., Ltd. and Aluminum, Ltd., Montreal, Canada, are planning a joint study of the suitability of sites in British North Borneo and the Gold Coast for aluminum reduction plants.

* In Canada the supply of suitable mine labor is again becoming very tight and the situation is expected to become more acute this fall.

* A drilling contract has been awarded for a minimum of $14,000 to the Atkins-Walker Co. of Duluth, Minn., for drilling into manganese-bearing iron ore on the Cuyuna Range on the boundary between Crow Wing and Aitkin Counties, Minn.

* Synthetic rubber is fast becoming the preferred raw material for the rubber industry with its use promising to surpass natural rubber both in the immediate and long-term future according to Harry E. Humphreys, Jr., president of the U. S. Rubber Co.

It's Everyone's Business

CHAIRMAN DOUGHTON of the House Ways and Means Committee announced on September 23 that the committee has scheduled public hearings on an excess-profits tax, beginning on November 15, 1950. In the meantime, the staff of the Joint Committee on Internal Revenue Taxation and the staff of the Treasury Department are to confer jointly on this question in the hope that such conferences may result in the preparation of a report on excess-profits tax proposals which could be used as a basis for testimony before the committee. It is thought that this procedure may aid in the draft of an excess-profits tax bill for introduction in the House when Congress reconvenes on November 27.

Congress wisely decided to delay consideration of excess-profits tax proposals until after the November elections. Legislating such a bill before the recess, when members of Congress were anxious to go home, could not have led to the creation of an equitable excess-profits tax. However, the decision to take up the subject at the "lame duck" session beginning November 27 is not much better.

There are, of course, some irrefutable moral arguments in favor of minimizing excessive profits in periods of defense preparation. However, caution must be exercised not to apply a tax on these profits blanket-wise. Certain justifiable exemptions from the tax are apparent and should be considered. Domestic corporations engaged in the mining of certain strategic minerals such as antimony, chromite, manganese, nickel, platinum, quicksilver, sheet mica, tantalum, tin, tungsten or vanadium should not be subject to the tax to the extent that their net income is attributable to the mining of these minerals.

The above minerals were exempt from excess-profits taxes during World War II when the economy was geared to full-scale war production. Failure to again exempt them, and others the Munitions Board may designate, would only result in hampering to an unknown degree the domestic development and production of these indispensable materials for defense. Our national security position can ill afford any maltreatment in this regard.

The Department of the Interior has taken over authority originally delegated to the National Production Authority in the Department of Commerce to administer priorities over: 1—nonferrous metals and minerals and ferroalloys processed through the smelting and refining stage and 2—ferrous metals and minerals, up to but not including the blast furnace operation. Trouble had been brewing between Interior and Commerce over the question of priorities even before the Defense Production Act was signed. Things became quite warm in early September when the Commerce Department would not allow James Boyd, Director of the Bureau of Mines, to sit in on a conference between Commerce officials and leading copper producers. The final outcome of the dispute is cause for rejoicing in the mining industry.

Since the Defense Production Act became law the mining industry has been waiting for the issue of Department of the Interior regulations governing the manner in which exploration, development and increased production of essential metals and minerals will be facilitated. According to information available at this time, it will still be a few weeks before these regulations can be presented and implemented.

The Reconstruction Finance Corporation has been authorized and directed by Presidential Executive Order 10161 to make loans for ". . . the exploration, development, and mining of strategic and critical metals and minerals." In addition, the General Services Administration has been authorized and directed by the same Order to "purchase and make commitments to purchase metals, minerals, and other raw materials, including liquid fuels, for Government use or resale. However, to date neither of these agencies has issued regulations pertaining to this authority.

Section 709 of the Defense Production Act provides that any rule or regulation issued under authority of the Act must be accompanied by a statement revealing that in the formulation thereof consultation has been effected with industry representatives, and that consideration has been given to their recommendations. However, such a statement may indicate that consultation with industry is impracticable or contrary to national security interests. In this connection it may be difficult for the Department of the Interior to state that it is impracticable to consult with industry in the formulation of their forthcoming regulations. The Department has a fine advisory group on minerals in the National Minerals Advisory Council. The Council has indicated that either its Executive Committee or the newly created Committee on Organization and Coordination of the Minerals Industry for Mobilization is ready at any time to meet with Interior officials on behalf of the entire Council on any matter affecting the industry. It is fervently hoped that the Interior Department will avail itself of this opportunity to receive the counsel of experts in the drafting of its regulations.

The Military Renegotiation Policy and Review Board of the Department of Defense on August 11 issued a list of materials, contracts for which are to be exempt from renegotiation. The list, which may be modified from time to time, includes various forms of each of the materials so exempted as follows: crushed stone, gravel, sand, alumina, bauxite, coal, copper, iron, lead, monel ore, nickel, and crude oil.

Manpower

The National Minerals Advisory Council, at the request of the Department of the Interior, has established a Committee on Labor to gather information on critical occupations in the mining, milling, smelting and refining industries. Specifically, information obtained is to cover the following points:

(1) A list of critical occupations at mine, mill and smelter and a brief description of each.

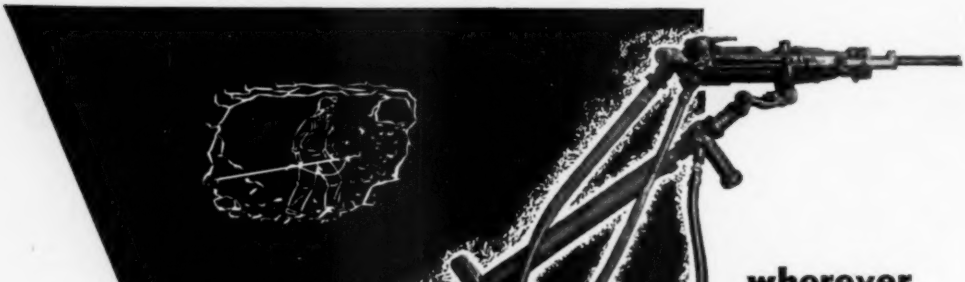
(2) Time required to train a man for a specific occupation.

(3) Current and recent labor situation in each category listed.

(4) Estimate of additional manpower requirements.

(5) Possibilities of replacing skilled men with older men and women in critical occupations.

Each of the members of the new Labor Committee —all industry representatives—has been assigned a section of the country in which to gather the information requested. The data to be presented to the Department of the Interior at the earliest practicable date so that the Department will be sufficiently informed and able to substantiate claims deferment claims for men in critical occupations when dealing with the Labor, Commerce, and Defense Departments.



FL2 Air Feed Leg Mounting an S48 Sinker

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NEW GARDNER-DENVER FL2 AIR FEED LEG MOUNTING

Small headings — sub-levels — hard-to-reach stopes — can now be drilled faster — with the new Gardner-Denver FL2 Air Feed Leg Mounting for sinker drills. Weighing only 42 pounds* — the FL2 is easily carried into remote workings — gives your runners these time-saving advantages:

— **faster setup**— easily and quickly attached to the drill.

— **easy collaring**— with stoper-type controls.

— **longer steel changes**— when used with carbide bits.

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— **extra long travel**— three feed travel lengths available: 36", 48" and 60". Extension leg adds an extra 24" for deeper holes.

*42 pounds for 36" feed travel.

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45-pound companion to the FL2

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— simple, effective steel puller — easily operated by foot or hand, assembled without wrenches.

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Direct Versus Indirect Control of the Economy

FACED with placing the nation in a state of partial mobilization, it is expedient that it be done in such a way that the national economy and productive capacity are strengthened. The immediate adoption of direct controls such as fixed prices and wages, rationing, manpower controls, and allocations conceivably could destroy the basis of our free economy. Controls breed more of the same and since the present program is a long term project, it should be undertaken within our present business framework so far as possible or until total war makes it impossible.

Production can be increased by a longer work week, private capital expansion, procurement from abroad, elimination of waste and featherbedding, no strikes, government production subsidies, government loans for or construction of war plants, and by diverting manpower, materials, and capital from unnecessary government projects. Shortages in materials like steel can be relieved gradually by indirect consumer controls.

With full employment and higher wages because of competition for labor and government purchasing there will be an inflationary trend. Nominal increases in prices will have some desirable effects such as eliminating marginal consumption as well as encouraging economical use of materials. Price control would, on the other hand, revive black markets. However, inflation must be avoided. It will raise the cost of the armament program, thereby increasing the national debt. Inflation feeds itself because labor demands for wage increases put more money in circulation and continue to drive prices up.

The Federal Government can do the most in curbing inflation. Curtailment of consumer credit restricts purchasing power. Steps have already been taken to do this. Commercial bank credit and mortgage credit must also be curtailed. Proper management of the federal budget so as to operate with a minimum of debt is imperative. This, of course, means increased taxes. A Federal Reserve System survey shows that for the year 1949 about 80 pct of the national income after Federal taxes was in the hands of the $7500 and below income group. This means that in broadening the tax base this group must be tapped more extensively. The Federal debt must not be financed by borrowing from commercial banks or the Federal Reserve System but rather from public savings. In addition to adopting a sound monetary policy the government must exercise economy in spending.

We are not preparing for a total war effort but for a prolonged period of military build up. We can maintain our present business structure and meet the demand on our productive facilities if the above policies are carried out and understood by business management, labor unions, and the American people. Direct controls may be required should an emergency arise, but they will have a better economy on which to work and, after the emergency, the transition to a peacetime economy will be easy. The National Defense Act is a beginning but it has not made clear whether we shall rely on direct or indirect controls, as both have been adopted. We must act to see that Congress and the Administration are not led too early into a rigidly controlled economy.

Jumbo and crew ready to drill round

Haiss ½ cu yd clam shell filling bucket

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Sinking

Tennessee Copper's

by L. WEAVER

THE Tennessee Copper Co.'s mines are in the southeast corner of the state of Tennessee, Polk Co., in the well-known Ducktown copper basin. Their new circular production shaft will eventually be the only ore hoisting shaft, and is located for handling ore from Boyd, Calloway, Mary, and Cherokee mines. A circular, rather than a rectangular, shaft was decided upon because: 1—The circular shaft section is stronger when lined with concrete. Reinforcing steel was not required, thus reducing lining cost by one-quarter. 2—Water sealing was desired; it was thought it could best be done with a concrete lining followed by pressure grouting to eliminate water seeps. 3—The shaft labor saving equipment could best be used in a large opening, so the round section was made for economy, sinking, mucking, concreting and water sealing. Less shaft steel is used than could have been possible with steel sets. There are no posts required; center beams and Tee plate assembly placed at 10 ft hold the 100 lb steel guide rails.

The shaft is 12 ft 7½ in. ID of the concrete walls. This is large enough for two 10-ton hoisting skips and a ladder compartment. One of the skip compartments is for hauling equipment.

The grouting equipment was a 2½x10-in. Gardner-Denver grout pump, air powered and developing 1000 lb pressure. A grout mixer, also air driven, fed the mixed solution into a tank coupled to the pump suction. Grout materials used were mostly water and Portland cement. Some Bentonite was used to seal porous ground near the surface.

Drilling

Five days, Monday through Friday, the drill, mucking, and maintenance crews sank five 6-ft headings for a total of about 30 ft. The concrete form was based on muck blasted by the Friday evening shift. Concrete was placed by the Saturday day shift; forms were moved up and concreting completed on Sunday. Forms remained in place on the second or top ring until another sinking cycle was completed.

One foreman, six drillers, and a sinking hoist man composed the drill shift. The crew plumbed the shaft and marked outside holes for size control. The foreman with two drillers did the

Circular Shaft

Surface bin and bucket in dump

plumbing, blew out sockets and checked for missed holes while the other four drillers readied the drilling equipment. The jumbo was then pushed over the shaft on rails and swung under

Mr. Weaver is superintendent of mines, Tennessee Copper Co., Copperhill, Tenn., and an AIME member.

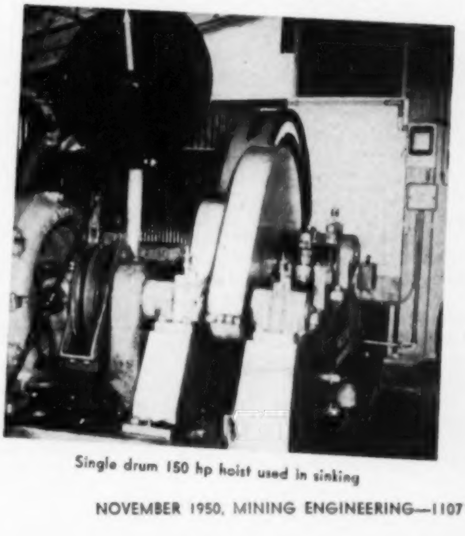
the sinking bucket by 4 chain slings hooked to the rim of the bucket and then lowered to the bottom for drilling. This work usually required about 1½ hr.

A jumbo was developed for this job which combined the principles of both wagon drill and tunnel jumbo. Four adjustable legs permitted leveling. The drills were mounted on a 3½-in. pipe frame. All had water and air hose permanently attached, with 2-in. air and 1-in. water hose used for connection to shaft feed lines. The four drill arms were hinged to permit adjustment in length and for folding into the frame for movement through the shaft.

An Ingersoll-Rand 4-in. D505 drifter drill, operated by two men, was mounted in the center of the jumbo and used to drill the cut holes consisting of at least six 3-in. diam burn holes and two 1⅝-in. blastholes. The material through which the opening extends is a tough rock, interbedded graywacke and chlorite schist; hardness and abrasiveness is classed as medium. Penetration per steel bit use is about 2 ft. A 4-point Timken type H thread, 1⅝-in. tungsten-carbide insert bit was used for drilling all of the holes except the 3-in. cut holes. The carbide bits cut about 300 ft per bit. Bits were of 1⅝ in. diam with gage increment in five changes of 0.002. These bits were tightly fitted on the drill steel, which was made up for 2 ft changes in lengths of 3, 5, 7, 9, and 11 ft. Bit sizes were put on steel in favor of drill hole gage and the bits were not removed during drill period.

Four Sullivan L-57 drills on 30-in. sliding cone beds with power feed were mounted on the swinging arms from the corner posts, and used for drilling the remainder of the holes. One man operated each drill and had 12 holes to drill. A

Concrete spouted from truck to 1½ yd bucket



Single drum 150 hp hoist used in sinking

total of fifty-six 7-ft holes were used. Ordinarily no water was found, so the round was drilled, the holes blown out and surplus water lifted into the ore bucket by a small centrifugal pump.

The average charge was about 200 lb of 45 pct semi-gelatin, wired in parallel and detonated electrically from the surface by 220 v.

Mucking

One foreman, two clamshell control men, one clamshell hoist man and one sinking hoist man were in the mucking crew.

The mucking hoists were located on the surface until the shaft had reached a depth of about 900 ft. Then they were moved to a station about 750 ft below surface where they remained until the job was completed. The single drum hoist was powered with a 150 hp electric motor and has a rope speed of 400 fpm. The hoist was equipped with a post brake, a motor shaft brake, depth indicator and a model D Lilly controller. A 1 in. non-spin hoist rope is used. Two Ingersoll-Rand 430 cu ft air compressors supplied air power.

The surface shaft door completely covered the shaft. Operated by an air jack, it was closed for loading and unloading men and supplies from the bucket, for dumping buckets of muck or water, and on many other occasions as needed. The shaft was ventilated by a surface fan which is rated to produce 2800 cfm of air through 2000 ft of 12-in. pipe. The ventilation pipe was a light weight 12-in. tubing anchored at the concrete and kept within 50 ft of the bottom.

A ½ cu yd Haiss heavy duty clam shell was operated by two Ingersoll-Rand air powered 4KUL hoists with a rated rope pull, at 90 fpm, of 3500 lb for mucking. The clamshell was laced with a short length of ⅝-in. cable with clevis on it for attaching to one of the hoist cables. The cables were wound on drums after the mucking cycle. This minimized the time used in bringing the clamshell to surface after use and taking it back for mucking the next round. The first mucking shift loaded twenty to thirty-five 2-yd buckets and the second shift completed the mucking and maintained service lines.

Concrete Lining

In effect the shaft length was concrete lined with a series of rings covering about 90 pct of the shaft area. The concrete mix was 1-1.46-3.07 (by weight), resulting in a strength of 5000 to 5600 psi. The ring of concrete was not reinforced and averaged about 1 ft thick. The concrete showed no damage from blasting, although the bottom of the concrete was about 9 ft above the rock, and had only about 55 to 60 hr setting time.

One leader, three timbermen, a surface hoist man, a pipeman and a supervising engineer foreman formed the concrete shift.

The crew first cleaned up fly rock, ventilated and sprayed the muck pile, leveled the top of the muck pile around the circumference for proper steel form footing. A 4-in. air line, 1½-in. water line and a 3-in. emergency pump line were then extended. These steel pipes were enclosed in the concrete as placed, to extend 6 to 8 in. into the muck and each was capped so service could be obtained from the connection at top of last concrete ring, preventing dirt or concrete from getting into the pipe. Then the forms were lowered into place, wedges put in and when in acceptable alignment the forms were braced securely on 4 sides with 2x4-in. pine lumber. A portable work platform was mounted near the top of the forms. The forms were then ready for placing the first concrete ring.

The concrete forms are 14 ft high, made of ⅛-in. steel plate reinforced with bent angles and 5-in. channels. A 7½-in. detachable wedge is held in place by bolts and the form is pulled open or closed by four 2-in. bronze threaded rods, ends of which are threaded right and left hand. The nuts for these bolts are anchored to the form on each side of the wedge gap. Maintenance of these forms was nil. Since two rings were required for the average weekly advance, on the second concrete ring a 5½-in. wedge was used because the bottom of the form was expanded in the concrete of the previous placement.

The steel concrete form was suspended with four ½-in. steel cables equally spaced around the circumference of the inside of the finished concrete. This location of cables kept the form in alignment when moving. The positive support was insurance against accidental dropping of the form, and having the form thus supported released main man-hoist and bucket for workmen to expedite movement of the form.

The four form cables extended from form to surface collar 10-in. sheave wheels, thence across the shaft collar edge to four other alignment sheaves which put the cables close together for clamping. These four cables extended to four reels where sufficient cable was available for reaching the completed shaft bottom.

A form hoist was located far enough from the shaft collar sheaves to permit a travel length of 50 ft. The hoist had a 16,000 lb rope pull at a speed of 8 fpm and a single drum with a reversible motor and brake.

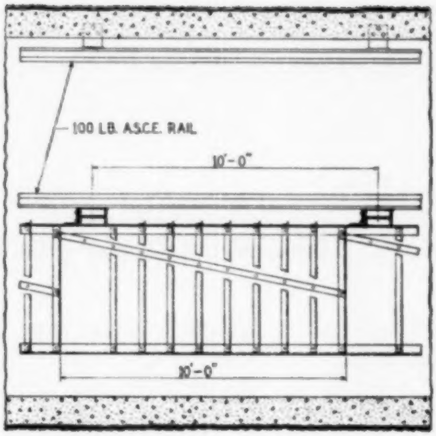
The hoist pull rope was 1 in. diam and fastened to a manifold clamp for attachment to the four form cables. A fixed clamp held tension on all four cables while the manifold clamp was being adjusted for another movement of the form.

One leader, three timberman assistants, one hoist man, two mixer plant operators, and two truck drivers were on the concreting shift.

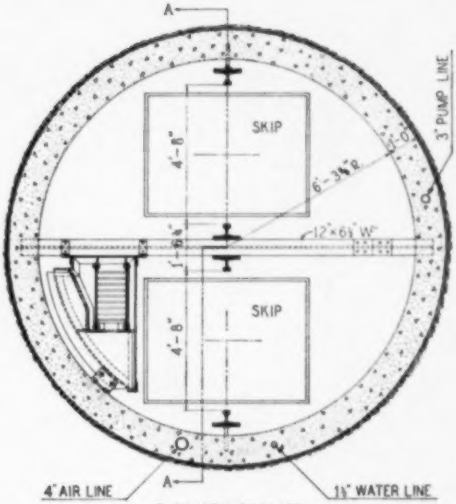
Trucks hauled mixed concrete about three miles from Copperhill to the shaft in 1½-yd batches on a schedule of about three loads per hr for each truck. The concrete was spouted into a 1½-yd bottom dumping bucket, which was pushed to the side of the shaft. The bucket had a swinging spout about 15 ft long under the discharge door. This spout was swung to the form by men on the temporary stage for placing the concrete. The concrete bucket doors were controlled by small steel cables long enough for the operator to reach from the portable platform. Four cable slings suspended the platform from the top of the steel forms.

This crew completed form preparation, placed the concrete, and cleaned all of the equipment in an 8 hr shift. The 28 to 30 yd of concrete were usually placed in 3 to 4 hr.

The next crew took out the form wedge, closed the form, hoisted it 13½ ft, leaving a 6-in. mini-



Above, vertical section A through shaft



PLAN VIEW OF SHAFT

mum lap, extended it, placed a 5½-in. wedge, leveled up and then placed concrete. This was done in about 3 hr, with 3 to 4 hr required for placing concrete, leaving about 1 hr for moving stage to surface and cleaning up the concrete bucket, platform and spouts.

An emergency air powered hoist to be used in event the electric hoist failed handled a completely enclosed cage, which swung free on a non-spin rope. It was lined up for travel near one side of the shaft, and when concrete was being placed it was lowered to the elevation of the top of the steel form. While a concrete bucket was being loaded and returned the two men working on the bottom platform stepped into this cage for protection.

The first lift of this shaft to 1400 ft is adequate for at least twenty years. The relatively shallow depth of the shaft and time available eliminated the need for paying a premium for speed at the expense of quality or first cost. An economical cycle was decided on which would average about 100 ft per month, including delays due to sealing out any water found.

Work was started in December, 1948. A temporary steel headframe and housing was erected, the 150 hp single-drum hoist, two 430 cfm compressors, electric power lines and station with all of the equipment used in sinking was purchased and installed in December and January. In February the shaft was extended to water level and the concrete lining was completed from bottom to surface. The total advance was 73.5 ft.

The total cost of sinking equipment installed was \$54,802.54. Salvage credit was \$17,900.00, leaving a total cost of \$36,902.54. The cost per ft on the 1411 ft of shaft was \$26.15. The sinking expense to March 1 on the 73.5 ft of shaft, including diamond drilling, engineering work, concrete lining and grouting, was \$23,072.25 and \$313.91 per ft.

Footage for the 9 months, March through November, 1949, was 917.5 ft, which is very close to the scheduled 100 ft. The total delay caused by water for the period was 6 weeks.

Sinking in October and November was not delayed by water. Six shaft rounds were lost from sinking as crews were breaking off level stations and moving hoists to an underground location.

The total sinking days were 37 for the two months, footage advance was 255 ft, making a daily average of 6.89 ft and a monthly average of 127.5 ft.

Cost Summary for The Period

Feet	Ft per Round	Expl Cost per Ft	Lb Expl per Ft	Ft per Man Shift—Total Crew	Yd Loaded per Loading Shift
255	6.89	\$7.11	29.4	0.212	35.92

Cost per Ft		
Sinking	Concrete	Total
\$82.43	\$36.12	\$118.55

Average Concrete Costs

Concrete to Shaft Cost	\$12.70 per yd
Moving forms and placing concrete	10.23 per yd
Miscellaneous expense	3.18
Total	\$26.11

The average yards of concrete used per ft of shaft was 1.55. The shaft was sunk open to bottom by lining walls with concrete. The final job was installing steel for supporting skip guides, ladderway, etc.

The plan and view of shaft in section shows type beams used, with the center I beam anchored in hitches cut in the concrete and the Tee plate supports placed in a slot cut by drilling and after alignment they were grouted in place. These sets were placed on about 10 ft centers.

The hitches were cut from a special shaft jumbo, starting from the top. When hitches were cut the steel was placed from this jumbo, starting at the bottom, finishing on the surface. The guides were then placed, starting at the surface and going down. The ladder was installed last. The cost of the shaft complete with ladder and guides ready for hoisting was \$225.57 per ft.

Cost Summary to Complete 1411 ft. of Shaft

	Cost per Ft
Preparation Cost	\$26.15
Sinking Cost	99.86
Concrete Lining	40.49
Grouting for Water Sealing	18.48
Steel, shaft sets, guides, and ladderway	40.59
Total	\$225.57

How to Operate

by Howard H. Fields

Any mining engineer with a desire to operate independently, with some financial backing, and with no fear of heavy responsibility and long hours, should be able to make a comfortable living in Mexico. Barring a few inconveniences such as a lack of satisfactory liability or accident insurance, the difficulty of obtaining repair parts for vehicles, and the length of time required to receive returns from shipments, the operation of a small mine is not at all difficult. It is, in fact, a little easier to find a good prospect in Mexico than in the U.S.

Mr. Fields, an AIME member, is an independent operator in Carbo, Sonora, Mexico.

The details begin with entering Mexico legally. This requires a "*Visitante*" passport that will permit the holder to work as an investor. It is good for six months, requires proper letters of character, identification, and usually, some assurance of financial stability. It is limited to a certain area —two or three states, and to mining operations. This type of passport costs $2.10, but often it requires a *mordida* to speed it up. At the end of six months it is possible to obtain an *Inmigrante* passport, which costs $102, is good for one year, and is renewable annually for 5 years, at which time a permanent passport is issued.

To discover commercial ore is as difficult in Sonora as anywhere else. On one's arrival in a mining town, word is rapidly spread and all sorts of mines are presented. It is easy to reject many of these, but the major portion require some thought and usually a trip. Mexican miners are optimists, usually lack assay sheets or smelter settlements, and carry all the details in their heads.

To visit prospects, it is necessary to carry grub, bedding, water, rope, etc., and to get underground to see their possibilities. The usual sampling equipment is necessary and the sampling should be done personally. One should also be ready to assay most of the samples brought in.

After hand sampling indicates a possible profitable operation, it is often desirable to ship a truckload to one of the local buyers for a real sample. Burros and trucks are available in most districts for such sampling and some times all or a large part of the cost of investigating a prospect can be covered by these shipments.

Location of Claims

If the prospect has an owner, an option can be taken either under the name of some reliable Mexican, before the *Inmigrante* passport is obtained, with an agreement to transfer later, or under the engineer's name after the *Inmigrante* has been obtained, but also after permission is obtained from the Sria. de Relaciones Exteriores in Mexico City. Care should be taken to be sure that the document is an *option* not a *contract* to purchase, because under the latter a tax of 10 pct plus becomes due immediately.

If the prospect is on open ground it can be lo-

a Small Mine in Sonora, Mexico

Old-style transportation

Building mine road

Typical camp at prospect

cated under the same personal restrictions as under optioning. There are two types of location: 1—*Cateo*, where there has been no mining at all, in virgin ground. One can locate 9 Hectares, about 22 acres, 300 meters square with sides N.S. and E.W. This costs 30 pesos and requires no monuments on the corners, merely a hole 3 meters deep, and two monuments in the center. Compass bearings to some prominent landmarks to locate it are included in the description. A *Cateo* requires no annual work— is not transferable, and is good for 2 years, at which time it is *Caduco* or vacant, unless it is changed to an *Explotacion* location. 2—An *Explotacion* location can be made any size or shape and have any direction. Its cost is 60 pesos for 10 Hectares, plus one for each extra hectare. After its acceptance it must be surveyed and monumented, with stone and mortar monuments, 1 M x 1 M x 1 M, so each one is visible from two others. Annual work must be done and reported and taxes paid to hold this type. It is never patented but can be held indefinitely in this manner and is transferable. An *Explotacion* location can be made covering new or old claims or changed from a *Cateo* within 2 years.

After a location is made and before the title is received, no shipping can be done until after receiving a *Permiso Previo*. The local mining agent can issue one in the case of a *Cateo* but that for *Explotacion* must come from Mexico City and are slow and hard to obtain. *Explotacion* titles can be expected about 6 months after location.

There are no restrictions on prospecting, development, and spending money with no production. Once production is imminent, the operation as a company or personal, should be registered in the Office of Hacienda (Treasury) in the locality that is home for the operation. This registration also covers the National Chamber of Commerce. An official set of books should be opened by a regular accountant who will handle the income tax matters. These books are only relative and have little bearing on the mine operation books because they will not admit all the matters and in the manner the local books show. This sounds most formal but is really simple and not costly. *Facturas* or sales tickets are required on all formal ore sales to smelters or mills.

By making a monthly statement to the Sria. de Havienda covering tons, metal content, and to whom shipped, a reduction in production tax can be had. A new mine receives 50 pct reduction for 2 years, 30 pct for 1 year, and 10 pct for 1 year. A mine that has been idle for 10 or more years receives the same, except 50 pct for only 1 year. This production tax is based on the gross value of any metal paid for. There are two other taxes called Ad valorem, which total 17.85 pct of an arbitrary value of the metals, but this is offset by a *bonificacion* or credit, which almost eliminates this tax, making it of small effect.

A recent decree gives a small operator a subsidy on production taxes, depending on his tonnage, i.e.: 80 pct reduction if less than 50 tons per month, 50 pct reduction if less than 100 tons per month, 30 pct reduction if less than 150 tons per

month. This is easier to obtain but cannot be had in addition to the first type.

It has been found to be desirable to have a personal representative in Mexico City. There are a number of excellent men who have friends in the various departments and can be most helpful in all sorts of matters. This representative usually is on a job basis and is not too expensive.

Labor

The basis of employment at the beginning is most important and will affect the labor question during the life of the enterprise.

Each man should be individually hired and sign a contract, say for 90 days. This contract states the work is the beginning or development of a mine and is not permanent. This should be renewed and kept in force. There is a chance that after several years in some labor dispute they might be questioned, but they are the best protection known for a small operation. Without such a type of individual contract, workers are entitled to 90 days severance pay when the mine closes. It is very difficult to discharge an employee but absence without permission and drunkenness are the most available causes. If one becomes too bad, discharging him and paying him a small sum is the usual out. A newly employed man works 30 days under test. If he is acceptable he signs a contract, if not, he is discharged with no obligation. The closer the scrutiny the foreman uses in sorting out the bad ones during this period, the surer the work will go smoothly.

Once a prospect seems worth developing, the problem of how formal a job is warranted has to be met. There is no general rule that can be applied. In Mexico where a peso is worth 11.2¢ and hand miners are paid 6 to 8 pesos a day, hand labor can often be used to good advantage in place of mechanized operations. In ordinary rock, hand drilling is as cheap as air drilling and produces cleaner ore. Windlassing can be done

The normal method is to use hand drilling and windlassing following the ore. Hand drilling is done with 4-lb hammers, ⅞ in. steel, tree branch spoons using frayed ends to extract the drillings, and 40 pct powder. Normally, a driller can drill 72 to 96-in. of hole per 8-hr shift. Ore is moved in rubber tired wheelbarrows up to 200 ft, dumped on shoveling sheets at the shaft. The usual windlass has an 8-in. barrel of pine, and a 1-in. hemp rope is used. Reinforced carbide cans make buckets and ladders, reversed, form skids. Carbide lamps are used and safety hats are required. When the mine shows more than normal promise, change is made to air drilling, and a gasoline hoist for a more formal development, with some blocking out of ore. Small operations usually stay close to the ore and do not carry on crosscut tunnel explorations.

When air is used light mounted wet sinker drills with ⅞-in. hollow hexagon steel, and automatic rotated wet stopers, with the same steel, are employed. After trying detachable bits, carballoy bits, hand sharpened steel and throwaway bits, the latter are now used with excellent results. A hand shanking device gives good results.

The varying types of ore bodies encountered in small mines, which usually are never really developed, force the use of only a few types of stoping. In narrow veins, open stopes with stulls and lagging platforms to reach the back are most common. When chutes, cars and tracks are installed, the mining is becoming formal. However, whenever possible, ore is broken apart from waste and the waste used for fill. Methods like shrinkage, etc., involve the tying up of too much ore for a small operation. The high cost of dimension timber limits its use, and local mesquite, palo fierro and scrub oak make fair stulls, which are recovered and reused whenever possible.

Due to cheap labor, ore sorting is a most important item in the small mine operation, and doubly so when a long haul to the market exists. There are many combinations of preparing the ore for shipment, but the most effective is screening over a wire ore screen of proper size with the shipment or rejection of the fines and the hand cobbing and sorting of the oversize. Each sorter has an old mortar block, hammer and a steel claw. This is a two-prong rake, made from short ⅞ in. hand steel, the prongs are 3 in. deep, 6 ft long, and it has a handle 8 in. long of the original steel. The prongs are separated about 2½ in. The sorter uses this claw in place of his fingers to turn over the oversize.

Typical specimens of sulfide ore and carbonate ores have been assayed and the oversize sorting is controlled by these specimens. Any change in the type means another assay and holding back some uncertain ores until the assay is available. A quick approximate assay of combined lead, zinc, sulfide, can be made by taking several type samples—crushing, cutting in two parts, assaying one set and grinding representative samples of the second set—taking a measured amount by volume and shaking it up in water in a narrow cylindrical bottle. Mark with a file where the line of separation occurs on the bottle for each type sample and from the assay of the corresponding part of each one each line will have an assay value. Taking the same amount of the unknown and putting it in the bottle, treating it the same, the line of separation will give the approximate assay.

Similarly, carbonate, carbonates and oxides of lead can be assayed approximately, at the mine. Prepare the samples in the same way as the sulfides, assaying one half of the type samples—grind one half, take a measured amount by volume, pan it carefully, retaining all the concentrates but eliminating all the waste possible. Then add a little saturated solution of commercial sodium sulfide, which coats all the lead except anglesite, black, as it coats it with a film of lead sulfide. Transfer these to a similar type of bottle and mark with a file where the lead sulfide shows separate from the waste. From the assay results of the corresponding parts of the samples, one knows the assay represented by each line. Taking the same amount of unknown and treating it exactly in the same manner, the line of separation gives you the approximate assay. It is surprising what good control this gives on lead carbonate ore. The regular assay office is complete with coarse and fine crusher and pulverizer and gas furnace, with usual balances and equipment for wet work.

In general, ore is hauled in trucks with flat wood beds about 12x7x2 ft. These are light, flexible and have a low center of gravity and short

wheel base. Usually they have two speed axles or a second transmission for the steep grades. The usual truck, Chevrolet or Ford, 1½ ton class, brings 4 to 4½ metric tons. Dual wheels, 8.25-20 and 9-20, 10-12 ply tires, off the road type, with deep corrugations and soft rubber are the best.

A larger GMC, 2 ton class, brings 7+ tons, but is a large investment for the average Mexican trucker. On a typical haul, 60 miles grade in favor of the load, the truckers receive $37.50 MN or $4.35 U.S., or $0.0725 per ton mile. Gasoline costs about 18¢ a gal, truckmen receive $15 MN a day. 9.00-20 tires cost $110 U.S. delivered in Sonora, and last 15,000 to 20,000 miles. The trucks are loaded and unloaded by hand, the standard price being $1.50 MN for each operation. The shipper pays for loading and unloading.

At the mine a restaurant is maintained, also lodging for the truckmen, who pay for their meals, an arbitrary $2 MN. A road crew is maintained to keep the road in good shape to warrant the low hauling cost. This crew consists of 10 men who use shovels, picks, rakes and a railroad drag. To reach one mine, a truck road was built through mountain terrain for 12.5 miles at a cost of $52,000 MN or about $600 U.S. per mile, using pick and shovel.

Markets

In choosing between U.S. and Mexican markets, there are two handicaps shippers to the U.S. have to overcome. First, the duties on lead and zinc ores of ¾¢ per lb on total metal content, and second, Mexican duties on exported ores are based on total metal content, in place of recovered values when shipped to Mexican plants. This is a very serious handicap.

Market for carbonate ores in the U.S. is logically the AS&R Co. plant in El Paso, but they are diverted to their Selby plant at times.

In Mexico the AS&R San Luis Potosi plant, and the Penoles plant in Torreon, Coah., are markets. Freight rates are generally a little higher from northern Sonora to Mexican plants but the duty handicap makes silicious carbonate shipments more attractive shipped to the Mexican plants. There are metallurgical reasons at times that offset these differences, but usually Mexican plants offer a better market except from northern Sonora. A sample or complete analysis should be submitted for rates before shipment.

Lead, zinc sulfide ores are harder to market, but there is a market in the U.S. at the Eagle Picher mill at Sahuarita, Ariz. The AS&R mill at Charcas, San Luis Potosi also accepts this ore. Both plants offer fair recoveries but the duty handicap on U.S. shipments, plus lower milling charges makes Charcas the more attractive on most ores. In either case, a sample must be submitted before shipment can be made.

Copper ores are logically shipped to Cananea, who are exceedingly interested in silicious ores containing over 55 pct silica. Favorable rates are offered for such ores containing gold, silver and/or copper values. Ores carrying less than 50 pct silica are not sought though are accepted in small quantities. Freight rates to Cananea are very low, silicious treatment rates are low and metal payments good, so there is no question about marketing this type of ore.

All plants are willing to guarantee freight on shipments from responsible shippers, both in Mexico and the U.S. On shipments to plants in Mexico the plant handles all the tax matters, while on exported ores a customs broker must arrange and handle the Mexican taxes and the U.S. import duties.

The supervision in general is under the owner or operator, who does the surveying, mapping and most of the geology, though outside help is brought in when a real geologist is necessary. The owner also looks after financing, ore sales, general labor and buying. A young American engineer helps in surveying, mapping, and in general machinery supervision, and is almost indispensable. The local accounting and handling of money is done by a cashier-bookkeeper, who makes weekly trips to the mine as paymaster, and who handles all the bookkeeping and cost accounting and writes the letters in Spanish. The mine foreman handles hiring, firing, directs mine operations, keeps time, dispatches ore trucks, receives supplies, does the local approximate assaying, and makes the daily report. He has a night foreman who handles a night shift under his direction. In addition, there is an assayer, competent to determine gold, silver, lead, zinc and copper. Also at the mine is a policeman to keep out bootleggers and maintain general order. A cook runs the boarding house and two chauffeurs operate a truck and pickup for general errands. This completes the overhead.

The great difficulty in small mine operation is to know when and where to end overhead. Costs must be recorded to operate intelligently but a small mine can bog down in paper work.

At one operation where the payday comes weekly, the foreman keeps the time in a regular time book. The bookkeeper-cashier in the town office types a formal payroll, goes to the mine, extends it there on payday, paying in cash and getting each man's signature on the payroll sheet. These sheets are sent to the official bookkeeper each month, because a small income tax is collected from each worker who receives over $6.66 MN each day. On a weekly basis at a small mine, it is easy to avoid any padding.

The foreman makes a daily report, covering working faces, men in various jobs, powder, fuse caps used in each place, each shift, cars or ore and waste moved. He also gives each truck as loaded two leaves from a triplicate book, which carry a number, estimated weight, class of ore and name of truck and driver. On arrival at the railroad, the truck is weighed in loaded, and out light, on truck scales, and these weights are recorded on both tickets, one going to the office and one is kept by the trucker until he is paid. A record of all weights is also kept.

The ore is unloaded on a ramp or platform the height of a boxcar floor, and when a carload (50 metric tons, is the usual shipment) has arrived, it is loaded by hand by contract, for $65 MN a carload. A shovel sample is taken, one in ten, crushed, coned, quartered to about 150 lb. It is then brought to the assay office, crushed, mixed and by proper stages brought to pulp size for assay. Before the mine operation became well established, a sample of the individual trucks was often taken to control or check the mine. The car loading sample checks the smelter and mill satisfactorily.

1950

Modern safety methods in British coal mines include the all-important dust suppression techniques to guard against pneumokoniosis.

1850

Britain's first Coal Mine Inspection Act was passed. An early attempt at accident prevention was Sir Humphrey Davy's safety lamp, several models of which (shown at right) are now in the Royal Institution in London.

British Mark a Century of Progress in Coal Mine Health and Safety

by V. S. Swaminathan

This year, Great Britain is looking back over a century to August 14, 1850, the day when the first "Act for the Inspection of Coal Mines" was passed in that country, an act which signaled the end of over two hundred years of hitherto uncontrolled and dangerous mining practices.

It was back in the time of Queen Elizabeth that thousands of pounds were invested in mining, particularly coal mining in England and Scotland. By 1640 there were collieries producing 10,000 to 25,000 tons of coal per year. In the seventeenth century the Tyne district coal industry was organized into powerful partnerships, and coal took an even larger part in industry, as a source of power and as fuel for the iron industry. In the early 19th century a "phenomenal, almost devastating expansion" characterized Britain's coal industry, and it was sparked by a 111 pct increase in the population of England and Wales in the first half of the century. Coal production for those 50 years increased 470 pct and pig iron production was up 946 pct for the period 1801-51.

The first public action on safety in coal mines came in 1812, following an explosion in Durham in which 92 lives were lost. It was then that Sir Humphrey Davy brought forth the safety lamp bearing his name, but for reasons then not understood the lamp did not come up to expectations. Serious explosions continued to occur, resulting in various government commissions which made needed recommendations, but legislation was not forthcoming. Britain was shocked, however, by what some of these commissions saw, particularly the use of plentiful child labor, including the sight of young girls working in the mines at some of the most dangerous tasks. The Royal Commission on Coal Mines wrote in 1842: ". . . instances occur in which children are taken into the mines as early as four years of age,

Mr. Swaminathan is assistant professor of geology at Presidency College, Madras, India.

sometimes at five and between five and six, not infrequently between six and seven and often from seven to eight, while from eight to nine is the ordinary age at which employment in these mines commences, . . . female children begin work in the mines in several districts at the same early age as the males . . . the regular hours of work for children and young persons are rarely less than eleven, more often they are twelve." A law subsequently passed in 1842 prohibited employment underground of females and boys *under ten years old.*

Although this last law established the principle of government inspection, it was never carried out in practice. Two more major disasters within three years, in which 135 lives were lost, gave impetus to an aroused public opinion which resulted in passage of the inspection law of 1850. The original law provided for four inspectors, and six more were appointed five years later. Further legislation has added to the staff, and now the law calls for 188 inspectors.

Progress made is brought out by salient statistics. In 1873 when 143 million long tons of coal were raised from all mines under the Coal Mines Registration Act, fatal accident rates were 2.08 and 7.5 per thousand persons employed and per million tons raised. In 1948 when the comparable output was 202 million tons the corresponding rates were 0.63 and 2.31.

In other words, the 1948 output was 59 million tons more and the persons killed 601 less than in 1873. On the basis of the number of people employed, or the number of man shifts working, with the single exception of the Netherlands, Great Britain now has the lowest accident rate, measured on any basis except output, of any coal mining country in the world.

Accident prevention work goes on in full force, attempting to improve mining conditions, using the tools of education and training, discipline and propaganda, and looking to protective equipment to solve some problems.

The Teepol injector, developed for use with Shell Chemical's Teepol wetting agent, is proving its worth in British coal mines.

The progress of mechanization, with all its benefits, has, strangely enough, accentuated the problem of pneumokoniosis. From 1931 to 1948 over 22,000 workers in British coal mines were certified as suffering from either silicosis or pneumokoniosis. In addition to this toll in human misery, a direct loss in working efficiency caused by diminished visibility due to dust may be gathered from the fact that in some pits the dust created simply by hewing is thick enough to make a miner's lamp invisible at arm's length.

Automatic water sprays are of course used in an attempt to suppress dust, and in a number of pits seam infusion, or the injection of water under pressure through cleavages in the seam, has been practiced. The extensive use of water for dust suppression in mines has proved dangerous, softening the floor and endangering the stability of pit props. Careful regulation of the moisture content of the coal is also necessary, as this content should not normally exceed 3 pct when the coal reaches the preparation plant. Wetting agents have been particularly effective in dust suppression programs, and extensive experiments under official auspices have proved the value of Teepol, a wetting agent manufactured by Shell Chemicals, Ltd. Hitherto the main stumbling block to the use of wetting agents has been the difficulty of maintaining a constant flow of the right strength solution at operating points. This has been overcome by the Teepol injector, making the use of Teepol solutions as easy as that of water. The normal strength of Teepol solution is 0.2 pct. By adding this amount of Teepol, the water is made wetter and the amount of water required is reduced by as much as 50 pct.

Wetting agents and water have also proved effective in the consolidation of deposits of loose dust on the floors of mine roadways. The National Coal Board's Scientific Branch has found the following technique satisfactory: for each 100 sq ft of road, four gallons of water containing 3 to 5 pct of Teepol in solution are used. The operation is repeated a few minutes later, and then a determined quantity of calcium chloride is added, three-quarters being laid immediately and the balance a few days later. The amount needed in a humidity of 60 pct is about 50 lb per 100 sq ft, reduced to about 25 lb when the humidity is 80 pct.

British mine inspectors have, during their first century of constructive work, built up a great tradition based on impartiality, mutual understanding, and respect, a specialized knowledge of mining dangers, and experience in a wide variety of mining methods and conditions. They have been aided in their work on an international scale by the I.L.O., which first took an interest in coal mine safety problems as far back as 1937. An I.L.O. committee drew up a model code of coal mine safety regulations before the war, and in September of last year a Tripartite Technical Conference met in Geneva, with respresentatives of 15 countries present. The model code, amended and revised by the Conference, was unanimously approved and submitted to the I.L.O. which will, in due course, authorize its distribution for the guidance of governments and the coal mining industry.

Caving and Drawing at Climax

A practical discussion of the theory of block caving is presented which applies particularly to the physical conditions of the Climax orebody although the conditions are sufficiently characteristic of caving orebodies to provide valuable information applicable elsewhere.

by F. S. McNicholas

UNTIL the fundamentals of the physical behavior of rocks are completely understood, progress in block caving must proceed upon a cut and try basis. Criteria of rock failure are many and varied. If a rock mass behaved as a homogeneous isotropic solid, the problem would be greatly simplified.

In practice we are never dealing with isotropic or homogeneous rock but with one that has been acted upon by geologic processes and which has pre-existing planes of failure. Under these complex conditions we do not find agreement. Actually there is little agreement between elasticians and physicists as to manner of failure in an isotropic homogeneous solid, although much progress has been made in the study of rock failure in recent years.

The following principles are expressed in simple terms which in the writer's opinion represent the present applicable horse sense reasoning which seems to explain caving action. Rock fails as follows:

1—By compression (crushing), 2—by shear, 3—by torsion, 4—by tension, a) straight tension, b) bending tension; 5—by attrition, and 6—by disintegration.

Rock is strongest in compression. Little advantage can be taken of compression to promote caving and fragmentation since weight cannot be put on entire area to cause crushing; putting weight on pillars in area causes only crushing of the pillars and transmits undesirable weight onto workings; weight develops too slowly to be utilized in mass progressive mining. Forces of compression can be utilized to aid other manner of caving and fragmentation.

Rock will fail in shear, usually over an undesirably large area, with poor fragmentation. Such failure is to be avoided in caving practice of strong rocks.

Rock is weaker in torsion than in compression or shear. No practical method is known to place rock in torsion. Some torsion naturally results from placing rock in unstable equilibrium. Torsion cannot be developed as a main method for caving ground.

Rock is stronger in straight tension than in bending tension. Rock is weakest in bending tension. Bending tension can be effectively developed by cantilever effect so such a method may be successfully employed in caving and fragmenting rock. Cantilever effect to fragment rock should be developed by a gradual continuous retreat of undercutting so the cantilever fails gradually from the end and not instantaneously through shear of the large overhanging block. In some weak ground it is necessary to avoid excessive cantilever effect as the cantilever may transmit undesirable weight to the adjoining area to be developed. For this reason at Miami caving by a panel system was changed to block caving.

Attrition of rocks as they move down to the draw points aids fragmentation. Disintegration of rock through chemical action, wetting or freezing occurs but is not important in fragmentation for mining purposes.

Mr. McNicholas, an AIME member, is a consulting engineer for the Climax Molybdenum Co., Denver, Colo., and for the Mufulira Mining Co., Northern Rhodesia.

In the stronger rocks cantilever effect is essential to secure caving with satisfactory fragmentation, unless the orebody is preweakened.

Where area is solid with no cutoffs caving is caused by removal of material from the base of the area, allowing natural forces to form an arch or cause collapse of the rock by shearing. If arching develops sufficient area must be drawn to destroy the arching effect through shearing of the rock. The strength of the rock determines the extent of area necessary to allow shearing. This condition allows free caving while arching is in progress, thus promoting caving in large chunks. In practice this is seldom done in the strong rocks. In weak rocks, such as at Miami, where arching effect is limited and cantilever effect to promote fragmentation is thought not necessary, this is the practice. Originally Miami used boundary weakening in a standard manner.

In the stronger rocks, where cantilever effect is necessary to properly cave and fragment the rock, boundary weakenings should be made to allow cantilever effect to be initiated and utilized to avoid failure of rock through shear, to speed up caving action, and to define the area desired to be caved.

Where cutoffs are made and undercutting retreats from the cutoff into the solid, strong rock (if amenable to caving) will cave arching into the cutoff with an overhang or cantilever, thus placing rock in bending tension. The same effect is secured when undercutting retreats from a boundary against caved material. If no draw is put on the area, broken material will flow under the cantilever and support it so it acts as a beam. When this occurs, the desired bending tension is removed and caving will perhaps stop. To secure the desired cantilever effect a small draw should be put on broken material under the cantilever to destroy beam effect.

The cantilever developed by a gradual continuous retreat of undercutting will place rock in bending tension and produce the utmost caving and fragmentation effect. Other forces of compression, shear and torsion will aid bending tension. Attrition as rock descends to the draw point will aid fragmentation.

Draw

The action of draw depends upon the material. In this discussion, the draw refers to Climax or similar ore which contains a majority of coarse chunks and a small amount of fines with little binder. This material does not consolidate to a marked degree over a considerable period. The draw under discussion is from completely filled areas.

The draw of this material affects an expanding column for a certain distance from the draw point. Many experiments on draw have established an angle of 82° as the average for this expanding column. However, this angle is the resultant of varying angles of draw and is true only for one vertical distance from the draw point.

The expanding draw of the ore column is flatter than this near the draw point and steepens with an increase in distance from the draw point.

Layouts should not be made which extend the expected draw on an angle of 82° for a considerable distance. Practically, the draw should be considered to be vertical from the limits which can be kept active around the draw point.

Practically, the maximum spacing of draw points should be controlled by the area which can be kept active by the draw. Greater spacing than this, as urged by economy of development, will result in poor or limited caving, irregular caving, funnelling, irregular draw, large dilution and low recovery.

The area which can be kept active depends upon the arching effect of the material which depends largely upon the size to which the material breaks.[1]

The maximum spacing is thus determined. The desirable spacing, less than the maximum, is determined by the cost of development and operation compared to profit from the variable recovery as related to spacing of draw points. Size of equipment, space required for the same, and tonnage demands influence this spacing.

The draw point spacing of about 34 ft centers, with 4 ft high x 8 ft wide fingers, is considered about right for the size ore being drawn, the equipment and methods used, and the general economics of the situation at Climax. This is about the maximum spacing that should be used regardless of size of material to be drawn.

The draw column has considerable supporting effect and if established in an undesired position has a tendency to maintain that position.

Boundary weakening or cutoffs to such an extent as to allow the draw to establish itself in this weakening and pull through to caved capping should be avoided.

The draw has a tendency to follow any line or area of weakness and may pull around and under a mass of ore, leaving this mass as an island which may not be recovered. This may happen when draw is from one finger over too extended a period.

When caving against caved capping on a boundary, draw must proceed from solid to the caved capping until 20 to 30 pct is drawn, at which time the entire ore mass is broken and probably of about the same density (fluidity) as the capping; so draw should then be vertical. A very limited draw, the reverse of this, to promote cantilever effect is desirable.

Regularity is most desirable. The draw is essentially irregular and effort should be made to keep it as regular as possible.

Any cutting of area to be caved into blocks or cutting it up by drifts or cross cuts is undesirable since this promotes caving in big chunks.

Development and mining should be planned and conducted in a systematic planned continuous retreat, mining all of the orebody to be mined by the initial mining and leaving no blocks, pillars or corners to be mined later. Retreat from the middle toward extremities of ore is desirable.

Cutting orebody into isolated blocks or leaving blocks for later recovery should be avoided. By cutting into isolated blocks or leaving blocks for later recovery, the benefit of regular orderly progression of mining is nullified, as is the benefit of cantilever action. More cutoffs are needed, draw control is more difficult and, if practiced, large tonnages of broken ore are unavailable. More area must also be opened up to produce the

same tonnage, and ore will break in a blocky form, while costs will be increased.

It is generally agreed that the spacing of diamond drill holes, as in the past, on about 400-ft intervals for preliminary information is still satisfactory.

Diamond drill holes on about 200-ft spacing is satisfactory for determining ore boundaries for mining purposes within the well established limits of the orebody.

In marginal areas, or when approaching the end of mineralized areas, or where other unknowns are suspected, diamond drill holes should be spaced to meet conditions.

An effort should be made to avoid fanning of holes. Holes should be, so far as possible, placed in a regular pattern to facilitate and improve accuracy in calculating grade. This means parallel vertical or horizontal diamond drill holes on regular spacing.

Layout Controls

Layout for caving ore (under capping) is subject to several controls. The angle of disturbance is established at 45°. This means the extension of a 45° line from the brow of a draw finger must pass over proposed adjacent slusher drifts or permanent workings or installation to avoid possible disturbance. Angle of cutoffs or boundary weakening on the footwall should be vertical or not flatter than 85°. On the hanging wall this angle is to be no flatter than 75°.

Where the dip of footwall is steeper than 45° the footwall area must be developed, mined and drawn from the top downward in blocks. The upper blocks are to be entirely drawn before the next lower block is drawn to an extent to endanger the upper workings.

The main draw must progress from the hanging wall to footwall, that is, from the solid toward the caved capping of adjoining stope, and be under strict control until the entire solid ore in the block is caved (except for the limited draw in reverse manner to promote cantilever effect). The initial draw should be small.

Where dip of footwall is flatter than 45° the area should be developed, mined and drawn in manner similar to the sill work with similar controls.

Still mining is to follow established spacing and controls. Development and concrete work should follow the established sequence of mining and be planned and prosecuted so as to remain in balance and continually provide area for stoping.

A tabulation, either graphic or by figures, or both, should be established and kept monthly to determine the progress of the work and the relative position of each class of work. This will show what class of work to push.

Ventilation development should precede mining so full benefit is secured and to avoid any possible question of unhealthy condition. Positive introduction of fresh air to each working place and positive evacuation of contaminated air through airways not used as manways is a must. Slusher operators in the new layout must be provided with independent fresh air supply. Accurate records of dust count to be kept to insure efficiency of ventilation and maintain satisfactory working conditions.

Standard slusher drifts shall be single ended with double fingers with a spacing of 6 car centers, or about 68 ft, on the sill.

On the footwall the spacing may be increased to a maximum of 75 ft, measured on the horizontal.

Fingers shall be on about 33⅓ ft centers on the sill or slightly more than this for the first set of fingers to allow proper installation of slusher hopper.

On the footwall, slusher drifts shall be offset horizontally, so ends do not connect, to allow scraping about 180 ft in one direction from a permanent hoist setup. This will allow a satisfactory ventilation installation. The turbulence caused when scraping ore into an ore pass gives a bad dust condition when scraper hoist is alternated from side to side. Good positive ventilation must be provided.

The characteristics of the Climax orebody, footwall and hanging wall are ideal for the full panel caving method as now practiced. Indications are that no preweakening of the orebody is needed. No change in the general method of undercutting is suggested. Detail of such undercutting is subject to continual experimentation.

Mining should be conducted to bring in stopes in the sequence established by the systematic continuous retreat method as planned. This sequence provides that stopes being undercut have caved material on not more than two sides.

Each stope should be blasted individually by blasting one or two rows of pillars at a time and retreating from the loose (cave) side to the solid. This should be done slowly to allow cantilever effect to develop. A slight draw is put on the area. Observation of remaining pillars will show when weight is developing and blasting should proceed.

Blasting several adjoining stopes with instantaneous removal of all pillars destroys to a large degree cantilever effect and promotes caving by shear in large blocks. The same is true to a lesser degree when all the pillars in one stope are blasted simultaneously. Such practice should be avoided where possible.

Complete cutoffs are essential on the footwall. On the hanging wall, partial cutoffs or boundary weakening at least half way to the hanging wall is indicated. The system of making a complete initial undercut retreating from the loose to the solid should develop cantilever effect in a desirable manner. While blasting in retreat for cantilever effect, put a light draw on the area to remove support from the cantilever, and destroy beam effect.

A light initial draw should be placed on the area being blasted to produce cantilever effect. The established draw should be followed. Draw should not exceed the rate of caving since to do so allows an open hole under the solid that allows free caving which promotes caving in large chunks. When drawing against caved capping on a boundary the draw to proceed from solid to loose with the fingers next the loose, lagging until 25-30 pct of area is drawn.

References

[1] *See F. S. McNicholas, V. C. Rogers and M. S. Walker: Analysis of Draw Point Spacing at Climax, and An Experimental Study of Caving and Drawing Large Orebodies. Trans., AIME, 1945.*

More Cost Estimates on Taconite

"The Taconites Are Ready", the editorial appearing on P. 933 of the September issue, has provoked comment from several informed engineers to the effect that the indicated profit margin was considerably exaggerated. The cost figures in the editorial, taken from the Congressional Record, were characterized as inaccurate by some correspondents. The theme of the story was, however, substantiated in the announcement (P. 1023, October) that Republic and Armco had joined forces in a $160 million project for taconite development.

Chief among the dissenters was E. W. Davis, Director of the Mines Experiment Station at the University of Minnesota. Mr. Davis has worked up the following set of cost figures:

Cost of Taconite Pellets Delivered to Lower Lake Ports

	Cost per gross ton of pellets	
Labor—1 man-hr per ton + social security, vacations, etc.	$1.65	
Power—70 kwhd per ton—1¢ per kw.	.70	
Supplies, Repairs, Maintenance, etc.	1.90	
Stocking and reclaiming pellets (winter storage)	.25	
Miscellaneous, including heat, insurance and contingencies	.20	
Total Operating Cost		$4.70
Taxes—all kinds	$ .20	
Royalties—Mine	.15	
Depreciation and interest—7½ pct x $20	1.50	
Total Overhead		1.85
Total Cost, f.o.b. mine		$6.55
Transportation to lower lake ports, including shrinkage		2.69
Total cost delivered to lower lake ports		$9.24
Lake Erie price, 64 pct Fe (dry), 62.5 pct Fe Nat. x $.15243		9.53
Profit per gross ton of 2240 lb.		$ .29

Books for Engineers

The Supervisor's Human Relations Pocket Library Kit. National Foremen's Institute, New London, Conn., 1950. 14 booklets. $3.50. A collection of booklets written for supervisors and designed to help them develop their ability to supervise people. Covers safety, planning, leadership, human relations, teamwork, etc. Manuals are pocket-sized, packed in a container.

Speaking Can Be Easy . . . for Engineers too. Engineers' Council for Professional Development, New York, 1950. 24 p. Gratis. A concise, practical approach to better public speaking and to better meetings. Hints on speaking, presentation, conduct of meetings, discussions, and preparation of publicity. Bibliography included.

Mineral Resources in World Affairs. Seventy-fifth Anniversary Volume, Colorado School of Mines, Golden, Colo., 1950. Heavy paper, 6 x 9 in. This volume is divided into eight numbers, containing the papers and discussions presented at the technical conferences of the 75th Anniversary program at the School. No. 1A—*Economics of the Mineral Industry*, 47 p, $.50; No. 1B—*Applied Geology*, 342 p, $3; No. 2A—*Petroleum Refining*, 162 p, $2; No. 2B—*Coal and Metal Mining*, 381 p, $3; No. 3A—*Metallurgy*, 60 p, $.50; No. 3B—*Petroleum Engineering*, 51 p, $.50; No. 4A—*Geophysics*, 103 P, $1; No. 4B—*Industrial Minerals*, 44 p, $.50. These numbers are bound in heavy paper, size 6 x 9 in.

Mining Year Book. Compiler, Walter E. Skinner, London, England. 1950, 810 p, $7. The international standard reference work on world mining companies for 1950. Detailed information on 950 companies the world over, 1,000 names and addresses of engineers, managers, etc. Includes sketch maps and a buyers' guide. This has been the standard reference book for over 64 years.

Atomic Energy Regulation. By Irving R. M. Panzer and editorial staff of Pike and Fischer. Albany, N. Y., 1950. 900 p, $25. A loose-leaf publication covering the legal and procedural aspects which stem from government regulation, domestic and foreign. Background material such as Senate hearings is given. Said to contain the only available collection of foreign statutes having to do with atomic energy. Includes: text of the Atomic Energy Act of 1946, procedure for acquiring radioactive isotopes, regulations on source material, production of fissionable material, uranium purchasing policies, and a bibliography on legal and procedural aspects of atomic energy. A six-month service plan provides pertinent new material, and is included in the original cost.

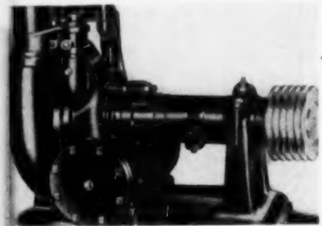
Selling to the Government. Chamber of Commerce of the U.S. 64 p, $.50. Explains procurement operations and lists agencies which make major purchases. Tells: who buys for the government, where to get the facts, what to do, and how the government buys. Sources of further information are listed, as are procurement laws and regulations.

Canadian Mines Handbook. Northern Miner Press, Ltd. 1950. 352 p, $1. Lists 8,893 companies, 803 in the active section, 8,090 inactive. Important source of information on Labrador-Quebec deposits, Noranda, Ontario, Newlund, Yellowknife, and other active areas.

Books may be purchased through I. P. Klein, Book Department, AIME. Whenever possible 10 pct discount is given. Order government publications direct from bureau concerned.

Improved Centrifugal Sand Pump

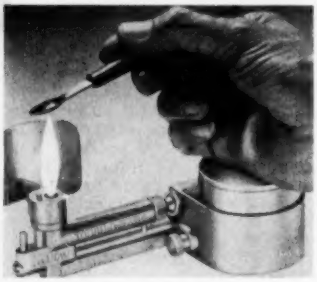
A new Model K heavy duty centrifugal sand pump is being offered by A. R. Wilfley and Sons, Inc. This model features 2-piece frame construction, permitting ready replacement of the rubber-lined or special-alloy intake chamber. An automatic check valve seals around the shaft when the pump is not in operation. This WILFLEY SEAL eliminates stuffing-box troubles, makes the pump especially adaptable to handling gritty pulps, slurries, sludges and hot solutions.



Features include easy removal of wear parts without disturbing intake or discharge piping, and ready adjustment of wear parts. Available in direct-connected, belt-driven, and overhead V-belt-driven types with 1, 1½, 2, 2½, 3, 4, 5, 6 and 8-in. discharge. **Circle No. 1**

Bunsen Burner Safe, Economical

An automatic Bunsen burner that flames only when needed, by touch control, is being made by the Hanau Engineering Co. Their Touch-O-Matic burner produces a full flame as hand or finger touches the on-off platform. When the hand is removed, the flame automatically



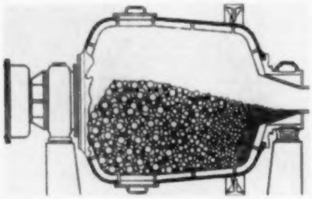
goes out. Thus the hazards of fire, burns, and fumes are eliminated, and gas consumption is reduced. The on-off platform also serves as a work-height hand rest. Any kind of gas may be used in the burner, including natural, manufactured, mixed, or bottled. **Circle No. 2**

Fault Finder Finds Cable Shorts

Joy Manufacturing Co. announces availability of a fault finder that permits quick location of short circuit and open-type faults in cable. This new unit requires no technical training or complicated calculations to operate. A lightweight transmitter and receiver are designed to operate on batteries and consequently require no external power connections. **Circle No. 3**

New Tapered Tricone Mill

The new "Tricone" ball mill for wet and dry grinding and pulverizing has been announced by Hardinge Co., Inc. The important feature is its slightly tapered shape, which keeps larger grinding balls at the feed end of the mill. Advantages claimed are: proper ball segregation, maximum energy gained at the feed end, convex heads increase ball turbulence, maximum working volume for minimum liner surface is provided, and



less wear on discharge grate as larger balls are kept from it by tapered barrel. **Circle No. 4**

U-Shaped Dozer Cuts Spillage

A new U-shaped bulldozer designed for use with the D8 track-type tractor has been added to the Caterpillar line. The new design allows long-haul pushing of loose material with minimum end spillage, and the U-shaped blade makes it a convenient tool for felling trees. Cutting width is 11 ft 11 in., and the blade is 45¼ in. high. The unit is cable controlled, consists of blade, push arms, trunions, cable, sheaves, and sheave brackets. The bulldozer has a box-type moldboard construction, 1-in. steel cutting edge, and heat-treated carbon steel end bits. It may be operated with the Caterpillar No. 24 front single drum cable control, or the No. 25 double drum cable control. **Circle No. 5**

Dollies for Mining Machines

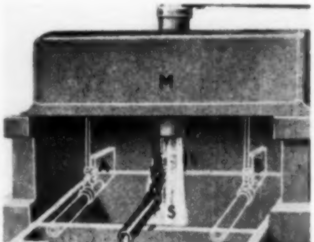
A series of streamlined "Phil Dollies" for hauling mechanical miners, loading machines, shuttle cars and other heavy equipment is being offered by Phillips Mine & Mill Supply Co. The 20-ton capacity model will handle the Joy Continuous Miner and all other equipment. A special 35-ton model will handle the Jeffrey Colmol. Requiring only 6⅛-in. head room, the dollies are attached to locomotives by means of a tow bar. **Circle No. 6**

New Blasting Galvanometer

An adjustable blasting galvanometer, which gives accurate resistance readings in ohms even when its activating cell is not at full strength, is now available to blasters. Made by the Atlas Powder Co., the instrument makes possible the checking of small resistances such as a single blasting cap. It may also be used as an ammeter in the detection of stray currents. The instrument is easy to use, having three terminal posts marked for connections in making three types of readings. **Circle No. 7**

Geary-Jennings Sampler

This mechanical sampler is the only sampler protected by patents which operates by taking a sample at right angles to the stream, thus insuring truly representative samples. It takes the whole of the stream part of the time, and takes equal percentages from all parts of



the stream. The following description is illustrated in the cut: cutter C remains stationary, out of stream, until mechanism M operates and moves cutter across stream S into position B. In traveling across the stream, C removes a cross-sectional slice of the stream. This is collected in any suitable manner. **Circle No. 8**

B&O Buys Largest Single Belt

The largest single conveyor belt ever shipped in a single roll is shown here after work on it was completed in the new $5 million belt plant of the B. F. Goodrich Co., Akron, Ohio. First major product of the new plant, the belt weighs 45,000 lb, is 48 in. wide, and the roll is 15 ft high. Designed by Goodrich for the Baltimore and Ohio Railroad, the belt will bring ore from ship to shore at the railroad's new dock in Baltimore Md. **Circle No. 9**

Jeep Mounted Prospecting Drills

Three lightweight Jeep mounted prospecting drills are available from Mobile Drilling, Inc. The B-27 Model augers 3 to 5-in. holes to depths of 75 ft, vertical or horizontal. The B-35 model has a range of 3 to 10-in. in depths to 100 ft. The "Jeep Jetter" is capable of 2, 3 and 4-in. holes to depths of 350 to 400 ft. All models operate from the power take off of either jeep, truck, or tractor and are said to be ideal for prospecting, sounding, soil sampling, blasting, cathodic pipeline protection, shallow water wells, seismographic work, etc. The Jeep Jetter is shown in the illustration. **Circle No. 10**

Fill out the coupon on the next page for more information about new products and literature mentioned here.

Manufacturers' Bulletins

11) OUTLET BOX: A new lightweight outlet box, providing five interruption-proof all weather outlets from one inlet, is described in brochure available from *Equipment & Service Co.* Each outlet will conservatively carry 1500 w and the inlet receptacle is rated at 6000 w, all at 115 v.

12) PERMALITE JOB DATA: Typical Permalite jobs with data as to the contractors, amount of Permalite and how it was used is contained in a 50-page booklet issued by *Great Lakes Carbon Corp.* Examples of its use in base coat plaster, lightweight insulating concrete for roof decks, and floor fills are also included.

13) EXCAVATOR: Booklet 400, released by *Marion Power Shovel Co.*, describes the machine, an all-purpose 1 cu yd excavator. Varied applications as a shovel, dragline, clamshell, crane, backhoe, and pile driver are illustrated. The machine is designed for use in drainage and reclamation, quarrying, coal and metal mining, and underground work.

14) ELECTRONIC INSTRUMENTS: New instruments for recording and indicating variables as temperature, resistance, pressure, and conductivity are illustrated in pamphlet W1821 offered by the *Bristol Co.* Reproductions of chart records, schematic drawings, and photographs of installations are also included. In addition, pneumatic and electric-operated automatic control models are mentioned.

15) BOW SAW: The Miner is a new type of saw designed for cutting timbers, logs and other rough work. Its narrow, tapered, frictionless blade cuts up to 1/3 faster than a one or two man saw. Literature may be obtained from the *General Steel Warehouse Co., Gensco Tool Div.* Handy tension lever permits quick changing of blades and high tension when in use. The frame is tapered at the point for work in close quarters and made of seamless, oval tubing for light weight and greater strength.

16) FLEXIBLE METAL HOSE: Catalog 500, issued by *Atlantic Metal Hose Co., Inc.*, contains the latest data compiled by the company's engineers. Section one includes flexible metal interlocking high pressure hose, and conveyor hose for ventilating and exhausting. The second section refers to seamless high pressure hose and diesel exhaust hose. Section three is concerned with metal lined gasoline hose and synthetic rubber gasoline hose. Test tables, bending diameter data, hydrostatic bursting pressures, and a series of installation diagrams are included.

17) REGULATORS: A 32-page booklet has been announced by *Spence Engineering Co., Inc.*, that contains information about pilot-operated pressure and temperature regulators. Case example problems are cited to help select the correct type and size regulator for particular applications. Main valves, controls and strainers are discussed and capacity and flow data is given. Thermostats, desuperheaters, electrically-controlled regulators, etc., are treated in detail.

18) SHAFT SPEED REDUCER: These vertical right angle reducers, described in bulletin 2110, available from the *Falk Corp.*, feature precision cut Herringbone double helical gears. They are available with sleeve bearings or roller bearings, depending up required applications. Vertical right angle units are equipped with a patented oil pump which assures positive lubrication at all times. Methods of selection, hp rating tables, dimensions, couplings, and torque capacity tables are discussed.

19) MINING CABLES: Increasing electrification of mines calls for constant improvement in the quality of mining cables. Latest improvements in cables, as well as rubber-sheathed trailing cables, and neoprene-sheathed cables, are discussed in 73-page booklet available from *Hazard Insulated Wire Works.* Illustrations are included of typical applications of portable mining cords and cables.

20) CONVEYORS: Data relative to *Industrial Engineering & Mfg. Co.'s* recent engineering developments on the line of industrial conveyors is presented in a new catalog sheet. A table is included that lists model number, channel length, frame width, belt width, and other pertinent facts.

21) CENTRALIZED LUBRICATION: Bulletin 484, issued by *Trabon Engineering Corp.*, describes this method of lubrication which is used on turret lathes and other machine tools, rubber and plastic mills, steel mill equipment, ore bridges, unloaders, coal and ore crushers, and conveyors and shaker screens. Every bearing gets its measured amount of lubricant because each feeder section is a miniature pump, operating in sequence with the other sections in a complete feeder block.

22) HOPPER DISCHARGE VALVES: Designed for continuous withdrawal of dust or powdered material from a collection hopper, these valves are illustrated in catalog page obtainable from *Buell Engineering Co.* The valve's capacity is 103.2 cu ft of dust per hr. The closing shock of the valve disc automatically shakes loose all clinging materials. This insures a steady flow, preventing any tendency to plug in the dust hopper.

23) NONFLAMMABLE FLUID: A technical bulletin describing Hydraulic Fluid F-9 is available from *Monsanto Chemical Co.* The fluid is suggested for use in items such as die casting machines, hydroelectric turbines, glass drawing machines and hydraulic presses. Physical and chemical properties and data on corrosion and stability are given in the bulletin.

24) SPROCKET RIM: These rims are designed for operation of overhead or inaccessible valves, and hopper devices from the floor. They eliminate the necessity of climbing on benches or ladders for the operation of valve wheels. A new folder describing the advantages of the device has been issued by *Babbitt Steam Specialty Co.*

25) HORIZONTAL SCREENS: These screens are built in single deck models ranging in sizes from 3x6 ft to 6x20 ft, while double and triple deck models are available in sizes from 3x6 ft to 6x16 ft. Pamphlet released by *Simplicity Engineering Co.* illustrates various models. They are used in heavy media separation, run-of-the-mine coal, gravel, etc.

26) CLEANERS: Oil bath air cleaners have proven their outstanding advantages where dust conditions are most severe. Booklet offered by *American Air Filter Co., Inc.*, describes the operation of Cycoil cleaner and silencer. A cross-sectional diagram shows the mechanism of the cleaner, and dimension tables are also included.

27) ROCK DRILL: Brochure issued by *Sandvik Steel, Inc.*, introduces Coromant, tungsten carbide tipped drill steel. It has made possible the development of a more efficient rock drill that drives better steel with less air and less heavy work.

28) PUG MILLS: Bulletin on pug mills for sintering plants and blast furnace dust catchers has been announced by *Wm. M. Bailey Co.* Single shaft mills and direct or rope drive double shaft mills are discussed. A thorough mixing of the materials and a fluffing action permits a higher bed on the sintering machine with the same wind box vacuum.

33) FURNACE REVERSAL: A recently developed unit for automatic control of openhearth furnace firing is completely described in catalog 4100 from *Bloom Engineering Co., Inc.* The average furnace can be completely reversed in 9 to 10 sec. Fuel savings, increased steel production, and greater flexibility in furnace operation are just a few of the advantages of the Bloom time cycle reversal system.

37) GRIPPING JAWS: *Instron Engineering Corp.* has released a catalog page describing and illustrating the various model jaws available. Resilient follow-up action, self-alignment of the gripping faces, and the positive tightness of the gripping surfaces in the direction of pull are three important features of the jaws.

38) DUST COLLECTION: The operation of cloth tube type collectors is described in an 8-page bulletin, "American Dustube Dust and Fume Collectors in the Mining and Metallurgical Industries," available from *American Wheelabrator & Equipment Corp.* These collectors are employed in recovering values in gases from roasters, sintering machines, induction furnaces, etc.

39) PRESSURE FILTER: Many applications of the Sweetland filter are listed, as well as structural details of the different sizes. Bulletin 114, obtainable from *Oliver United Filters, Inc.*, is well illustrated with photographic details of special features such as sight glasses and sluicing mechanism. General dimensions of the various models are given in a general data table.

40) FIRE BRICK: Information presented in a new pamphlet offered by the *Ironton Fire Brick Co.* is intended to aid in the selection of fire brick. These bricks are especially resistant to slag erosion, clinker action and abrasion at high temperatures. The pamphlet describes the use of Ironton's reliable airsetting bonding mortar.

41) IMPACT BREAKER: A new 8-page bulletin, IMP-1, tells the inside story of double impeller impact breaker action, available from *Iowa Mfg. Co.* It describes exactly how the breaker achieves the high capacity production of cubical shaped aggregate with low power requirements. Construction features are illustrated and described in detail and dimensions and specifications are also included.

42) SHOVEL: The Lodove 1-yd combination overhead and front end shovel for International Harvester tractors is fully described in illustrated catalog published by *Service Supply Corp.* It substantially increases loading production because turns are eliimnated; as many as 1900 per 8-hr shift substantially lengthen tractor life. Overhead loading steps up output as much as 50 pct.

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Some Applications of Millisecond Delay Electric Blasting Caps in Mining

by D. M. McFarland

A FEW years ago a novel electric detonator known as the split-second or millisecond delay electric blasting cap was introduced for use in quarry blasting. Regular electric blasting caps fired in series may be depended upon to fire within a millisecond or so from the first to the last in a series. Regular delay electric blasting caps are provided that fire one period after the other period in intervals of ½ to possibly 1½ sec. Most split-second or millisecond delays are designed to fire one period after the other period in possibly 25 to 50 millisecond intervals. The ear is not capable of detecting time intervals of this magnitude.

The primary thought at the time millisecond delays were introduced was to investigate the results on rock breakage by firing a line of holes in a quarry face so that charges in adjacent holes would not be detonated simultaneously. This could not be accomplished satisfactorily with regular delays. The time interval between successive periods of ½ to 1 sec was sufficient to permit considerable movement of the burden. If the burden of one hole was reduced to a great extent by the firing of an adjacent hole, the firing of the hole with the reduced burden would likely reveal this lack of confinement by a terrific report and wild throw of rock.

In the early blasts with millisecond delays it was observed that instead of the usual sharp report, the blast had a muffled sound and vibration was not as perceptible as when simultaneous firing was used. Because many quarry operators were being threatened with injunctions or suits for damages by neighbors who claimed structural damage to their buildings, millisecond delays were tried extensively in quarries. In the majority of these trials, the results were very satisfactory. The seismologists recorded the ground movement created by many blasts and verified the initial observations that millisecond delays could be used to reduce vibrations appreciably. In the past few years the advantages of this principle of nonsimultaneous firing of the charges in blasts has become generally accepted. Today the quarry operator who has vibration troubles, inadequate breakage, and excessive backbreak and has not investigated the possibilities of millisecond delay blasting is ignoring a remedy that has proved satisfactory for many. His complacency may be costing him money.

Because of the results attained in quarry blasting, it was logical that millisecond delays should be tried in construction work such as in road cuts. As formations in this type of work are likely to change rapidly with advance of the cut, it is more difficult to evaluate results than in quarry blasting. However, this improved control over timing has been beneficial in limiting throw, promoting fragmentation, and reducing overbreak. In blasting near buildings the reduction in vibration and in throw has been especially helpful. As blasters employed in construction work learn what may be accomplished by closer control over the time of firing of explosives charges, more and more millisecond delays are being used to supplant instantaneous electric blasting caps.

Improved Fragmentation Underground

With this background of promising results, it was not surprising that millisecond delays should go underground. In limestone mining use of millisecond delays as compared with use of cap and fuse or electric blasting caps showed improved fragmentation in stopes and in slabbing operations. Then an opportunity developed to use millisecond delays in some tunnels being driven in a limestone mine (fig. 1). Using the normal charge employed and merely substituting three millisecond delay periods for three regular delay periods, there was a noticeable difference in the appearance and the position of the pile of rock after a blast. A greater portion of the face was exposed, the crest of the pile was farther from the face, and the pile was heaped high along the center line of the tunnel leaving room to walk along the ribs to the face. Fragmentation was appreciably increased. It gave the impression that the slabs had been thrown against each other with tremendous force, promoting the movement of the broken rock along the center line of the tunnel away from the face. Because the drilling and the charge weights were unchanged, the evidence was convincing that the difference in timing was responsible for the difference in results. Probably a greater portion of the energy from the explosives had been expended in doing useful work on the rock. Zeros followed by two periods of millisecond delays were used in the V cut and in two slabs to either side of the cut in this simple round.

When millisecond delays, substituted period for period for regular delays, are first tried in a drift round in a mine, and the usual charge of explosives

D. M. McFARLAND, Member AIME, is Manager, Technical Division, Atlas Powder Co., Wilmington, Del.

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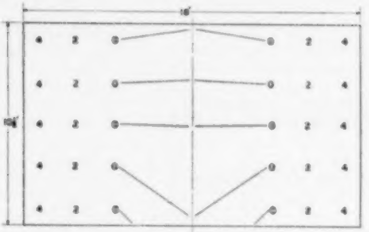

Fig. 1—Tunnel round in limestone mine. Numbers indicate millisecond delay period.

loaded in each hole, the user is likely to receive a shock upon returning to the face to observe results. The usual report is that the round pulled well, but the throw of the rock was excessive, thus creating a pile too shallow and too extended for efficient loading with a mucking machine. To avoid occurrences such as this that displease the miners and result in their opposition to any further trials, it is the practice of many mining companies to limit their experimental work to some unused section of their operations. When they have developed the new procedure to the extent that reasonably consistent and favorable results are obtained, trials in active operations follow.

If there is no opportunity to substitute millisecond delays for regular delays except in actual operations, in drifts as well as in shafts, experience has shown that it is advisable to reduce the usual charge of explosives for the first trial by at least one fourth. This will likely arouse considerable opposition from the miners, but it may avoid excess throw that might damage supply lines and timbers.

New Control of Timing

Although a review of the literature will give the impression that almost every conceivable type of drift round has been tried, it must be kept in mind that either cap and fuse or regular delay electric blasting caps were used to fire the charges in the holes. Except for pyramid cuts where the holes first to fire met at a common point or a V cut where two holes met at the apex, it was impossible to expect simultaneous firing of the cut holes when primed with cap and fuse. With the introduction of electric blasting caps it was possible to fire cut holes almost simultaneously and thus depend upon concerted action from them. However, after the cut holes were fired with instantaneous electric blasting caps, each of the successive delays that followed were expected to fire individual holes that were designed to break to a newly created free face. This holds true for delay electric blasting caps as well as cap and fuse detonation because the delays of any specific period cannot be depended upon to fire simultaneously. With millisecond delays those of any specific period do not fire simultaneously, but they do fire with much less variation in timing, one from the other, than do regular delay electric blasting caps. To utilize this newly acquired control over timing to the greatest advantage is the problem that concerns us. It is somewhat comparable to the problem which occurred when simultaneous electric blasting caps were first introduced on construction work and blasters started to use them in place of caps and fuse. It is likely that this change, which took place many years before my experience with explosives, presented many problems at the time. Now blasts numbering several hundred holes fired instantaneously are an accepted practice.

Control of Throw and Breakage

From experience with electric caps in shooting angles in limestone mines (fig. 2), the principle of accurate timing has been investigated. Assuming that all factors other than timing of the electric blasting caps are held constant, it may be shown that the less the variation in time of firing of the holes, the greater the throw of the angle. However, to obtain this great throw, fragmentation of the angle is decreased. Some systems of primary blasting in limestone mines require throw and tolerate poor fragmentation that must be corrected by secondary blasting.

When tunnel men of long experience insist upon using first period delay electric blasting caps in cut holes and not zero delays, or instantaneous electric blasting caps, they are following a procedure that our experience indicates was correct to reduce throw and to promote fragmentation of the cut.

One of our competent technical representatives had the opportunity to substitute millisecond delays in a tunnel where regular delay electric blasting caps were being used. Regular first delays were being placed in the cut holes to reduce throw and thus protect the timbering. When millisecond first delays were used in the four cut holes, a noticeable increase in throw was observed. This can be expected because four millisecond delays of the first period will fire much closer together than four regular delays of the first period. The next trial was with two first period millisecond delays in one pair of cut holes and two second period millisecond delays in the other pair of cut holes. This selection of delays in the cut holes reduced throw appreciably and was used in a number of rounds that followed.

In blasting a number of experimental rounds with millisecond delays, it was found possible to reduce the drilling by several holes per round and to reduce the total explosive charge appreciably. Fragmentation was reported to be improved and overbreak to be decreased. These two factors were responsible for a reduction in the cars required per unit of advance to transport the muck from the tunnel. A very apparent reduction in smoke and fumes after a blast was noticed. Although part of this may be attributed to the reduced charge, it is likely that a greater portion of the energy of the explosives was expended in doing useful work on the rock as indicated by less severe air blast as the rounds were fired.

In a potash mine where the face is undercut, there was an opportunity to study the effect of the selection of millisecond delay periods upon the movement of the burden. Essentially, past practice had been to shoot the line of holes adjacent to the undercut with

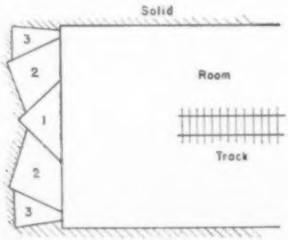

Fig. 2—Angle shooting in limestone mine. Numbers indicate sequence each angle is shot.

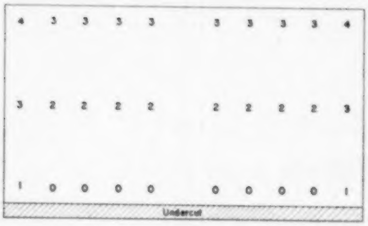

Fig. 3—Former type of round in potash mine, using regular delays.

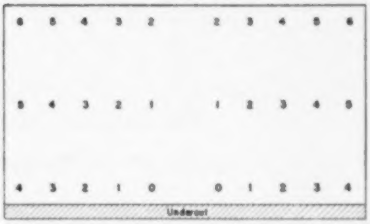

Fig. 4—Millisecond delay round in potash mine resulting in loose muck pile loading.

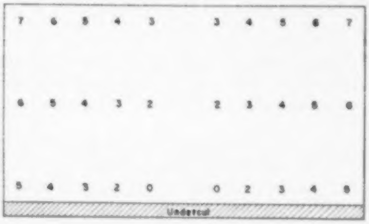

Fig. 5—Millisecond delay round resulting in tight, high muck pile.

zero delays with the exception of those at either end, called the rib holes, in which first period delays were used (fig. 3). The second row from the undercut was primed with second delays except for the holes at either end in which third delays were used. The third row from the undercut was primed with third delays with fourth delays at either end. Although millisecond delays had shown marked improvement when substituted for regular delays, the study was made to ascertain if breakage and digging might be improved further.

The preliminary results that were reported indicated that by using a delay pattern to cut slabs diagonally across the face rather than parallel with the undercut, slabs were reduced in size and breakage improved. For example, assume that there are five holes to either side of center in each of the three rows parallel with the undercut (fig. 4). The bottom row starting from center and progressing to the right or to the left would have 0, 1, 2, 3, 4; the second row, 1, 2, 3, 4, 5; and the third row, 2, 3, 4, 5, 6. This arrangement resulted in the center of the face being moved out farthest and provided a loose pile for the loading machines. When a period was skipped using 0, 2, 3, 4, 5 to either side in the bottom row; 2, 3, 4, 5, 6 in the second row; and 3, 4, 5, 6, 7 in the top row, movement of the face was less and a tighter pile resulted (fig. 5). This of course was not desirable. However, it gives us a clue on how throw of a cut might be retarded. This problem is far more complicated when millisecond delays are used than when regular delays are used. Although insufficient evidence is at hand to formulate any definite rules, from results to date we can infer that the smaller the time interval between the firing of the cut holes and those first to follow, the greater the throw of the cut is likely to be. Possibly this effect might be influenced to a lesser degree by the holes firing two periods after the firing of the first cut holes.

Test Procedure

Unfortunately, from this point on we must follow an uncharted course and by trial and error develop the basic information about how increasing or decreasing time intervals between millisecond delay periods in developing a cut in any specific formation affects results.

The logical procedure would be to drill and blast the cut holes alone when primed with zero delays, which are instantaneous electric blasting caps. If the throw is excessive, substitute first period millisecond delays for zeros in the cut holes. If throw needs to be reduced farther, then try zeros in the first pair of cut holes, first period millisecond delays in the second pair of cut holes, and second period millisecond delays in the third pair of cut holes where a six-hole V cut is used. By starting with a pair of first period millisecond delays, then a pair of second period millisecond delays followed by a pair of third period millisecond delays in a six-hole V cut, the use of any of the simultaneous firing zeros will be avoided. Extreme throw should not be expected.

After some satisfactory combination of delay periods has been decided upon for the cut, it might be well to investigate the effect of increasing time intervals when firing the cut holes plus the first holes to follow the cut holes. It is considered advisable to skip a delay period between the cut holes and the first ones to follow the cut. If this combination increases throw appreciably over the cut fired alone, then a longer interval between the cut holes and those first to follow might be tried. When a reasonably satisfactory combination has been found, it should be possible to complete the round by using alternate numbered millisecond delay periods. If for example, a six-hole cut is being primed with two zeros, two first delays, and two second delays, then 4, 6, 8, 10, 12, 14, and 16 delay periods may be used to complete the round. This will give a total of nine delay periods in addition to the zeros. As more information is developed on the use of millisecond delays in drift rounds, it is likely that simplified drill patterns will be developed that will enable the user to minimize the number of delay periods that are required. Some investigators have been exploring these possibilities and have published findings surprising to those well acquainted with theories that have been accepted in the past.

To make any prediction at this time as to the extent that millisecond delays will replace regular delay electric blasting caps one or two years from now in mining operations is merely a guess. However, the use of any product that at no additional cost will enable mining men to convert a greater portion of the energy from explosives into useful work is likely to increase rapidly. This has taken place in quarries and will likely follow in mines as the rules governing the effective use of fast delay electric blasting caps are developed.

The Influence of Certain Inorganic Salts on the Flotation of Lead Carbonate

by Maurice Rey, Paul Chataignon, and Victor Formanek

It is found when floating oxidized lead ores by sulphidization, that the presence of calcium salts in the water, is usually detrimental and lowers the recovery.

This effect is particularly marked in dry countries such as North Africa, where the waters often carry large amounts of calcium sulphate and where the ore may even contain gypsum.

The effect of calcium salts is readily visible. Whereas in their absence cerussite is quickly stained brown and then black by sodium sulphide, in their presence the mineral remains very light in color. A similar effect is produced when barium sulphide is used as a sulphidizing agent instead of sodium sulphide. Magnesium salts have little or no effect and even tend to reduce the detrimental effect of calcium salts.

A study of this phenomenon indicates that it is due to the precipitation of calcium or barium carbonate in contact with the mineral simultaneously with the formation of lead sulphide.

The chemical reactions can be interpreted as:

$$PbCO_3 + Na_2S = PbS + Na_2CO_3$$
$$CaSO_4 + Na_2CO_3 = CaCO_3 + Na_2SO_4$$

They might also be written:

$$PbCO_3 + S^{=} + Ca^{++} = PbS + CaCO_3$$

The precipitation of calcium carbonate can be followed by the lowering of the pH with which it is accompanied. Magnesium carbonate is more soluble than calcium carbonate and usually does not precipitate under the conditions prevailing.

It is interesting to note that calcium salts have no effect on anglesite (lead sulphate) because calcium sulphate is soluble, but barium salts hinder the sulphidization of anglesite because of the precipitation of barium sulphate.

Remedies

When calcium sulphate is present in large amounts, the softening of the water with soda ash is usually too expensive to be considered, but the precipitation of the objectionable calcium carbonate can be prevented in two different ways.

One is the use of sodium hydrosulphide instead of sodium sulphide. This salt gives a lower pH than sodium sulphide and does not bring about the immediate precipitation of the calcium which remains in solution as calcium bicarbonate.

The other procedure is to add ammonium salts such as the sulphate or chloride which have the property of increasing markedly the solubility of calcium carbonate. Ammonium salts have other effects such as cutting down conditioning time and accelerating flotation. They should be added to the flotation cells rather than to the ball mills.

Table I. Effect of Sodium Hydrosulphide and/or Ammonium Sulphate on the Flotation of Two Oxidized Lead Ores

Reagents	Mibladen Ore[a]		La Plagne Ore[b]	
	Concentrate Pb, Pct	Tailing Pb, Pct	Concentrate Pb, Pct	Tailing Pb, Pct
Without addition of $CaSO_4$				
Na_2S	56.0	0.68	43.4	0.65
NaSH	56.2	0.60	40.4	0.50
$Na_2S + (NH_4)_2SO_4$[c]	54.6	0.56		
$NaSH + (NH_4)_2SO_4$[d]			43.0	0.66
With addition of 40 lb/ton $CaSO_4$				
Na_2S	39.4	3.85	34.0	3.21
NaSH	49.1	0.70	49.6	0.88
$Na_2S + (NH_4)_2SO_4$[c]	53.9	0.85		
$NaSH + (NH_4)_2SO_4$[e]	51.0	0.45	41.1	0.78

[a] Mibladen ore, 7 pct Pb, 80 pct of which is oxidized.
Flotation of galena with 0.1 lb per ton amylxanthate.
Flotation of cerussite with 7 lb per ton Na_2S; 60 pct or the equivalent amount of NaSH. Four additions of 3.0, 2.0, 1.0, 1.0 lb per ton; and 0.5 lb per ton amylxanthate in two stages.

[b] La Plagne ore, 6 pct Pb, mainly oxidized.
Flotation of galena with 0.08 lb per ton amylxanthate.
Flotation of cerussite with 7 lb per ton Na_2S; and 60 pct or the equivalent amount of NaSH; 2 lb per ton sodium silicate; 0.32 lb per ton amylxanthate.

[c] 12 lb per ton. [d] 4 lb per ton. [e] 8 lb per ton.

In table I are given the results of representative tests on two different oxidized ores. They show that the strongly detrimental effects of calcium sulphate can be offset by the two procedures outlined above.

Sodium hydrosulphide is now used regularly on a mill scale on certain ores.

Tests are being carried out with ammonium salts.

It should be noted that malachite is subject to influences similar to cerussite.

One final word of caution—when the ore is rich in primary slime, which is flocculated by the calcium salts, it may be indispensable to remove these by washing or precipitation with sodium carbonate instead of keeping them in solution by the above methods.

Acknowledgment

We wish to thank Société Minière & Métallurgique de Penarroya and Minerais et Métaux, for permission to publish these results.

MAURICE REY, Member AIME, is Professor of Non-Ferrous Metallurgy, School of Mines, Paris, France, PAUL CHATAIGNON is Director of Ore-Dressing Laboratory, Minerais et Métaux, Paris, and VICTOR FORMANEK is Research Engineer.

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The Effect of Mill Speeds on Grinding Costs

by Harlowe Hardinge and R. C. Ferguson

Laboratory and plant data covering 12 different operations show that lower than "standard" ball mill speeds increase grinding efficiency. In the case of high pulp-level mills, the gain is so great that the increase in capital cost of the larger lower speed mill will pay for itself in less than a year's time.

THE object of this paper is to show the economic advantage of operating ball mills at relatively slow speeds. Although many operators know that grinding costs are reduced by operating mills at low speeds, they seldom do so for a variety of reasons. When capital is limited, first cost is of primary importance, and installation of the less costly, high-speed mill is favored. When materials are in short supply, as during a war period, capacity rather than economy is the primary consideration. Existing mills are then speeded up to provide the desired increase in capacity, and new mills are purchased without consideration of maximum grinding efficiency. When the mill speed is increased to secure more production, the increase in grinding costs is often obscured by the total benefit derived from the additional output. This increase in grinding cost is seldom recognized as being sufficient to be worth correcting. For these reasons one may be lead to the erroneous conclusion that slow-speed operation is only occasionally applicable.

The effect of mill speed on grinding efficiency has been studied in laboratory tests.[1,2] The results of some of these tests are given in tables I and II and fig. 1. In these tabulations, batch tests using ore charges of 125 to 200 lb were considered comparable to high pulp-level operation and charges of 50 to 100 lb comparable to low pulp-level operation. For the high pulp-level operation, these tests indicate the existence of a maximum in efficiency for operation at a speed of 50 pct of critical, both for the soft (dolomite) and hard (chert) ore. In low pulp-level operation, a decrease in speed increases the efficiency of grinding dolomite; no maximum of efficiency appears in the range of speeds tested. The results with chert were inconclusive.

If the performance of large mills is indicated

HARLOWE HARDINGE, Member AIME, is President and General Manager, Hardinge Co., York, Pa. and R. C. FERGUSON, Member AIME, is District Manager, Hardinge Co., Hibbing, Minn.

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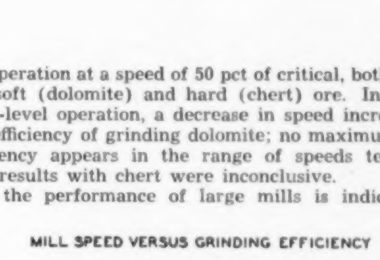

Fig. 1—Data taken from tables I and II, with 80 pct of critical speed as the base.

Table I. Speed vs. Efficiency in a High Pulp-level Laboratory Mill[a]

Speed, Pct Critical	Chert: Ore Charge, lb[b]	Chert: Surface Tons per hp-hr, Individ.	Chert: Surface Tons per hp-hr, Avg	Chert: Efficiency Gain, Pct[c]	Dolomite: Ore Charge, Lb[b]	Dolomite: Surface Tons per hp-hr, Individ.	Dolomite: Surface Tons per hp-hr, Avg	Dolomite: Efficiency Gain, Pct[c]
30	200	22.7	22.4	2.75	200	66.8	67.5	14.20
	150	22.4			150	68.2		
	125	22.1			125	67.2		
40	200	23.5	23.1	5.95	200	64.5	66.8	13.05
	150	23.2			150	67.3		
	125	22.6			125	68.5		
50	200	24.2	23.7	8.72	200	68.0	68.5	15.90
	150	24.1			150	69.5		
	125	23.5			125	68.0		
60	200	22.5	22.4	2.75	200	64.7	67.4	14.05
	150	22.5			150	68.5		
	125	22.2			125	69.0		
70	200	21.8	22.0	0.92	200	61.2	63.8	7.95
	150	22.3			150	64.8		
	125	21.9			125	65.4		
80	200	21.3	21.8	0.00	200	53.6	59.1	0.00
	150	22.1			150	61.9		
	125	22.0			125	61.7		

Data from TP 581, U.S. Bur. of Mines.

[a] Batch ball mill, 19 in. ID x 36 in. length; ball charge, 796 lb, 2½ in. maximum ranging down to 1 in.; ball volume, 45 pct; pulp density, 60 pct.

[b] Feed, 1.7 pct +8-mesh, 0.4 pct —200-mesh; product, the —200-mesh fraction ranged from a minimum of 13.1 pct to a maximum of 18.0 pct.

[c] Percentage gain in efficiency referred to the efficiency of the 80 pct critical speed run.

Table II. Speed vs. Efficiency in a Low Pulp-level Laboratory Mill[a]

Speed, Pct Critical	Chert: Ore Charge, Lb[b]	Chert: Surface Tons per hp-hr, Individ.	Chert: Surface Tons per hp-hr, Avg	Chert: Efficiency Loss, Pct[c]	Dolomite: Ore Charge, Lb[b]	Dolomite: Surface Tons per hp-hr, Individ.	Dolomite: Surface Tons per hp-hr, Avg	Dolomite: Efficiency Gain, Pct[c]
30	100	21.8	21.5	1.86	100	66.9	66.7	17.0
	75	21.4			75	66.6		
	50	21.4			50	66.5		
40	100	22.0	21.7	0.91	100	65.8	65.0	14.05
	75	21.6			75	65.8		
	50	21.4			50	63.5		
50	100	22.4	21.9	0.00	100	65.6	62.8	10.20
	75	21.8			75	62.8		
	50	21.5			50	60.0		
60	100	22.0	21.8	0.45	100	67.3	62.5	9.65
	75	21.8			75	62.0		
	50	21.6			50	58.1		
70	100	21.7	21.7	0.00	100	64.1	60.6	6.33
	75	21.7			75	60.2		
	50	21.7			50	57.5		
80	100	21.9	21.9	0.00	100	60.1	57.0	0.00
	75	21.9			75	57.1		
	50	21.9			50	53.6		

For source of data and meaning of [a], [b], and [c], see footnotes to table I.

Table III—Field Performance of Conical Mills.

Company Name	Cons. M & S Co. Kimberley, B.C.		A.S. and R. Co. Parral, Mex.		Hudson Bay M & S Co.		Nevada Mines Div. McGill-Kennecott Copper Co.	
Mill size	10x48	10x48	8x36	8x36	10x66	10x66	8x30	8x30
Capacity, tons per day	1,085	1,085	200	200	550	450	408	451
Installed hp	300	300	150	150	400	400		
Hp to operate	242	210	155	150	436	295	150	119.8
Mill speed, rpm	19	16.1	23	22.5	21.4	16	28	24.5
Pct critical	76	64	82.7	80.9	86	63	100	88
Ball load, lb	50,000	46,000	30,000	30,000	76,000	65,000	25,000	25,000
Ball sizes	3 in.	3 in.	3 in.	3 in.	3 in.	3½ in.	4 in.	3 in.
	C.1	C.1	C.1	C.1				
Lining con., #/ton	0.04	0.05	0.239	0.231	0.168	0.32[a]		
Ball con., #/T	0.85	0.55	3.45	3.44	3.06	2.3	1.55	1.34
Feed size	4-mesh	4-mesh	4-mesh	4-mesh	⅛ in.	⅛ in.	1 in.	1 in.
Pct plus	17.8	16.7	7.1	7.1	2.4	13.5	9	9
Product pct +48-mesh	2.0	4.6	3.7	3.4				
Product pct —200-mesh	53.5	55.2	60.2	60.9	84.8	90.0	45.7	47.8
Circulating load	5,065	5,915	920	1,200	3,000	4,000		
Tons —200-mesh/hp-hr	0.077	0.087	0.0336	0.0354	0.039	0.053	0.038	0.067
Remarks and Source	From File No. 289C. Also Taggart		Data from AS&R El Paso Office to Hardinge Co., 1937		[a]High wear reduced later by high lifters. Data H.Co. *O.R.* 533 and 583.		Data from Taggart	

Company Name	Kerr-Addison Gold Mines		Cons. Arizona Smelting Co.		Mufulira Copper Mines		Cananea Con. Copper Co.	
Mill size	10x72	10x72	8x36	8x36	10x72	10x72	8x28	8x28
Capacity, tons per day	697	724	132	140	854	830	300	300
Installed hp	450	450			450	450		
Hp to operate	492	462	114	101	390	385	110	92.6
Mill speed, rpm	21.8	19.8	18	16	19.5	18	25	17
Pct critical	88	79	65	59	78	71	67	59
Ball load, lb	76,000	76,000	28,000	28,000	75,000		16,000	16,000
Ball sizes	3½	3½			4½			
Lining con., #/ton	0.129		0.3	0.3	0.3			
Ball con., #/ton	1.75		2.41	2.41	2.5	2.0	6.3	4.6
							C.1	C.1
Feed size	⅜ in.	⅜ in.	1 in.	1 in.	¾ in.	⅜ in.	4-mesh	4-mesh
Pct plus	11	12	28	28	0.4		3.4	3.4
Product pct +48-mesh								
Product pct —200-mesh	60.8	64.0	52	52	57	65	26.8	27.8
Circulating load	2,450				3,620	8,000	Open Cir.	Open Cir.
Tons —200-mesh/hp-hr	0.031	0.036	0.023	0.027	0.046	0.049	0.0291	0.036
Remarks and Source	File 432K and H. Co. *O.R.* 693		H.Co. *O.R.* 62, 1920		H.Co. *O.R.* 617 3-10-39 5-16-34		H.C. O.R. 377 5-1-22	

Table IV. Effect of Mill Speed on Grinding Cost at Lake Shore Mines

5x16 Ft Tube Mill Dischg. Arrangement	Speed		HP		Ball Wear Lb per Day	Ball Cost Dollars per Day	Days for Liners	Cost of Liners Dollars per Day	Cost of Grates Dollars per Day	Total Cost Dollars per Day	Pct Cost per Day	Grind Cap. Rating, Pct	Cost + Capacity
	RPM	Pct Crit.	Output	Cost Dollars per Day									
Trunnion Overflow	30	84.6	182.5	18.25	421	15.55	1,300	0.77		34.57	107.1	123.5	86.8
	27	76.0	164.0	16.40	408	15.10	1,350	0.74		32.24	100.	122.0	82.0
Grate Discharge	30	84.6	204.5	20.45	590	21.80	1,200	0.71	1.50	44.46	138.8	147.0	93.7
	27	76.0	184.0	18.40	572	21.10	1,250	0.68	1.50	41.68	129.2	145.0	89.1

Data from *Trans.* C.I.M.M. (1940) 43, 427.
Tube mills in third stage of fine-grinding circuit. Classifier, overflow: 5 to 8 pct +40 microns (90 to 93 pct —325-mesh). Cast balls: ¾ in., 477 to 514 Brinell hardness, and $0.037 per lb in 1939. Liners grooved. Feed: sands from secondary tube-mill classifier, 200 to 300 tons per 24 hr (including circulating load), size approximately 5.4 pct +65-mesh. Mill load: 45 pct of mill volume for trunnion-overflow mill and 50 pct for grate-discharge mill.

Table V. Speed vs. Efficiency from Field Performance Given in Table III

Company Name	High Speed		Low Speed		Rate of Change of Efficiency, Tons —200-Mesh per Hp-hr per Pct Critical Speed	Rate of Change of Energy Required, Hp-hr per Ton —200-Mesh per Pct Critical Speed
	Speed Pct Critical	Efficiency, Tons —200-Mesh per Hp-hr	Speed Pct Critical	Efficiency, Tons —200-Mesh per Hp-hr		
(1)	(2)	(3)	(4)	(5)	(6)	(7)
C.M. & S. Co.	76	0.0770	64	0.0870	0.000833	0.1244
A.S. & R. Co.	82.7	0.0336	80.9	0.0354	0.001000	0.8407
Hudson Bay Co.	86	0.0390	63	0.0530	0.000609	0.2945
Nevada Con.	100	0.0380	88	0.0670	0.001417	0.9492
Kerr-Addison	88	0.0310	79	0.0360	0.000555	0.4978
Con. Ariz.	65	0.0230	59	0.0270	0.000667	1.0735
Mufulira	78	0.0460	71	0.0490	0.000428	0.1901
Cananea	87	0.0291	50	0.0300	0.000246	0.2352
Avg	82.84	0.0396			0.000719	0.5257

by the performance of laboratory mills, it may be assumed that for high pulp-level operation efficiency is at a maximum in the vicinity of 50 pct of critical speed irrespective of the hardness of the ore. For low pulp-level operation, increased efficiency of grinding with reduced speed may be assumed for soft ores and an indeterminate change in efficiency for hard ores. The effect of speed on the efficiency of large mills is shown in table III and summarized in table VIII. Table III lists only conical mills because data for these mills were available to the authors. These tabulations also show that circulating loads tend to increase as mill speeds decrease. Table V lists speed vs. efficiency from field performance given in table III. Data on cylindrical mills are not as readily available. It is reported[3] that in the operation of 9x9 cylindrical ball mills at Climax Molybdenum Corp. a speed reduction from 76 to 66 pct of critical was accompanied by an increase in efficiency of between 6 and 7 pct when the mills were operated without grates. When operated with grates, the same speed reduction showed only a 3 pct increase in efficiency. This tends to verify the laboratory results indicated in tables I and II. Gow et al[2] found an increase of from 19.1 to 27.3 surface tons per hp-hr (i.e., a 42.93 pct increase) when the speed of a 6x4 trunnion overflow cylindrical mill grinding dolomite was reduced from 80 to 60 pct of critical. The Lake Shore Mines, table IV, increased their 5x16 tube-mill efficiency 9.03 pct when the speed was decreased from 84.6 to 76 pct of critical.

Table VI. Speed vs. Ball Consumption From Field Performance Given in Table III

Company Name	High Speed		Low Speed		Change in Ball Wear with Change in Critical Speed, Lb per Ton per Pct Critical Speed
	Speed Pct Critical	Ball Wear Lb per Ton	Speed Pct Critical	Ball Wear Lb per Ton	
(1)	(2)	(3)	(4)	(5)	(6)
C.M.&S. Co.	76	0.85	64	0.55	0.0250
A.S. & R. Co.	82.7	3.45	80.9	3.44	0.0056
Hudson Bay Co.	86	3.06	63	2.03	0.0330
Nevada Con.	100	1.55	88	1.34	0.0175
Con. Ariz.	65	2.41	59	2.41	0.0000
Mufulira	78	2.50	71	2.00	0.0714
Cananea	87	6.30	59	4.60	0.0607
Avg			69.27	2.338	0.0304

To evaluate the effect of a reduction in speed upon the overall grinding economy, grinding costs and overall capital costs must be combined. Grinding costs are composed of the expenses for power, balls, liners, maintenance, and labor. Capital costs consist of the costs of the mill, motor, classifier, freight, and auxiliaries.

In the cost calculations shown in table IX, power to operate a mill at different speeds is assumed to be proportional to the rpm within the range of interest. Ball consumption is estimated as follows: the change in ball consumption per unit change

Table VII. Computed Power Requirements and Ball Consumption for Various Pct Critical Speeds

(1) Speed Pct Critical	(2) Tons —200-Mesh per Hp-hr	(3) Pct Increase in Tons —200-Mesh per Hp-hr Below 80 Pct Critical	(4) Pct Decrease in Tons —200-Mesh per Hp-hr Above 50 Pct Critical	(5) Hp-hr per Ton —200-Mesh	(6) Pct Decrease in Hp-hr per Ton, Below 80 Pct Critical	(7) Pct Increase in Hp-hr per Ton Above 50 Pct Critical	(8) Ball Consumption Lb per Ton	(9) Pct Decrease in Ball Wear Below 80 Pct Critical	(10) Pct Increase in Ball Wear Above 50 Pct Critical
82.84	0.0396			25.25					
80.	0.0416	0.00	34.18	24.04	0.00	51.96	2.664	0.00	52.05
70.	0.0488	17.31	22.78	20.49	14.77	35.84	2.360	11.41	34.70
69.27							2.338		
60.	0.0560	34.61	11.39	17.86	25.71	19.22	2.056	22.82	17.35
50.	0.0632	51.92	0.00	15.82	34.19	0.00	1.752	34.23	0.00

Table VIII. Summary of Mill Speed Data

(1) Mill Size	(2) Range of Speeds Pct Critical	(3) Pct Increase in Efficiency Tons per Hp-hr	(4) Avg Pct Increase in Efficiency Per Pct Decrease in Critical Speed	(5) Avg Pct Decrease in Ball Consumption per Pct Decrease in Critical Speed
19x36 in. cyl. mills	80-50	15.90	0.53	
5x16 ft cyl. mills	84.6-76	9.03	1.05	0.359
6x4 ft cyl. mills	80-60	42.93	2.15	
9x9 ft cyl. mills	76-66	6.50	0.65	0.000
Conical mills[a]	80-50	51.92	1.73	1.141

[a] Conical mill data taken from table VII.

Table IX. Comparative Costs at Different Pct Critical Speeds

	(1)	(2)	(3)	(4)
1	Mill size	10x10	10x11	10x12
2	Pct critical	80	60	50
3	Speed rpm	20.1	15.1	12.5
4	Ball charge lb	100,000	108,000	117,000
5	Pct of mill volume	41	40	41
6	Tph new feed	47.5	47.5	47.5
7	Ton—200-mesh per hp-hr	0.0416	0.0560	0.0632
8	Hp-hr per ton —200-mesh	24.04	17.86	15.82
9	Hp consumed	580	431	384
10	Tph —200-mesh produced	24.128	24.128	24.128
11	Hp installed	600	450	400
12	Ball wear lb per ton	2.664	2.056	1.752
13	Annual grind cost: power, in dollars	41,760.	31,032.	27,648.
14	Annual grind cost: balls, in dollars	59,221.	45,705.	38,947.
15	Grinding costs: power and balls, $	100,981.	76,737.	66,595.
16	Cost of mill and motor, $	58,000.	63,000.	68,000.
17	Mill cost plus 1 yr cost of power and balls	158,981.	139,737.	134,595.
18	Reduction in power and ball cost per year over 80 pct critical speed mill	0.	24,244.	34,386.
19	Saving in ball and power costs, cents per ton over 80 pct critical speed mill	0.	7.08c	10.05c
20	Reduction in ball and power costs per year using 9x9 mill data only	0.	9,109.	12,920.
21	Saving in ball and power costs for 9x9 mill in cents per ton	0.	2.66	3.77

Line 1 Mill sizes determined by hp requirements, line 10, and pct critical speeds, line 2.

Line 6 Determined by the average of 12.2 hp per ton of original feed for the mills shown in table III.

Line 7 From table VII, col. 2.

Line 8 From table VII, col. 5.

Line 9 580 hp assumed for illustration. 431 and 384 hp determined by multiplying line 7 by line 9.

Line 10 Line 9 x line 7.

Line 12 From table VII, col. 8.

Line 13 Based on a 7200 hr year power at $0.01 per hp-hr.

Line 14 Based on a 7200 hr year forged steel balls at $0.065 per lb delivered (1948 prices).

Line 20 Derived from table VIII, col. 4, by direct ratio. This assumes ball consumption is also altered proportionately $\frac{0.65}{1.73} \times 24.244 = \$9.109.00$.

in percentage of critical speed is computed in col. 6, table VI for each of the mills cited and the average of 0.0304 then calculated. By averaging cols. 4 and 5 of table VI, an average ball consumption of 2.338 lb per ton is computed for an average speed of 69.27 pct of critical. This is used as the basis for the calculation of col. 8 of table VII, assuming a linear relation between speed and ball wear with a slope of 0.0304. Liner costs were not evaluated because of the lack of quantitative information.

Table IX shows the comparative costs involved in grinding an ore at an hourly rate of 47.5 tons to produce 24.128 tons of —200-mesh per hr when mills operated at 80, 60, and 50 pct of critical speed are used. The final cost figures are arrived at without consideration of costs of liner wear, maintenance, labor, classifier, and auxiliaries. Table IX indicates a substantial saving in overall cost when low-speed operation is practiced. The gain is still substantial even if the 9x9 mill data are used. This mill, lines 20 and 21, shows the least gain of all mills tabulated except the laboratory unit.

Conclusions

1. Grinding efficiency increases as mill speed decreases within the range of practical operation.

2. Both power and ball cost per ton of —200-mesh produced decreases with a decrease of mill speed.

3. A slow speed, high pulp-level mill with sufficient additional volume to equal the capacity of an equivalent higher speed mill will make up the difference in capital cost between the two mills in well under a year's operating time through the saving in power and ball cost alone.

4. If an existing high pulp-level mill operating at "normal" speed is replaced by a new, but larger, lower speed mill of the same capacity, the new mill can pay for itself in less than two years' time out of the savings in power and ball cost alone. Only a very moderate resale value for the old mill is assumed in this case.

5. If an existing high pulp-level mill is replaced by a slow speed larger mill, an increase of 20 to 25 pct in capacity is possible without increasing the power consumption or cost of operation. If the decrease in operating cost per ton is also capitalized, this decrease alone will pay for the new mill in less than three years' time.

References

[1] TP 581, U. S. Bur. of Mines.

[2] A. M. Gow, M. Guggenheim, A. B. Campbell, and W. H. Coghill: Ball Milling. *Transactions* AIME (1934) **112**, 24.

[3] Private communication from E. J. Duggan, Mill Superintendent of Climax Molybdenum Corp., March 19, 1948.

Separation of Precious Metals from Anode Slimes by Flotation

by R. T. Hukki and U. Runolinna

Preliminary separation of precious metals by flotation can offer a simplification of the conventional method of treatment of anode slimes. Laboratory flotation experiments show that rich gold and silver concentrate can be obtained with excellent recoveries.

THE purpose of this paper is to present the results of an investigation into the possibilities of separating precious metals from anode slimes by selective flotation. The work was carried out at the State Research Institute in Helsinki, Finland. The sample of anode slimes tested was submitted by the Outokumpu Co. in Finland.

In 1947 the average assay of filtered and dried anode slimes in the Outokumpu refinery in Pori was as follows: Au, 0.50 pct; Ag, 9.38 pct; Cu, 11.02 pct; Ni, 45.21 pct; Pb, 2.62 pct; Fe, 0.60 pct; Sn, 1.00 pct; Sb, 0.04 pct; As, 0.70 pct; Se, 4.33 pct; S, 2.32 pct; SO_4, 2.17 pct; SiO_2, 2.25 pct. The daily assay naturally varies considerably from the figures presented above.

The present conventional method of treatment of anode slimes consists of the steps shown in the flowsheet in fig. 1. As seen from this simplified flowsheet, the present method of treatment involves many steps by which the components of the anode slimes are removed one after another until the main products, gold and silver, may be separated. Due to the small scale of operations, the above procedure is carried out in small batches requiring considerable amount of manual labor.

If, however, the anode slimes could be treated in

R. T. HUKKI, Member AIME, is Professor of Mineral Dressing, Finland Institute of Technology, Helsinki, Finland, and U. RUNOLINNA is Research Engineer, The State Research Institute, Helsinki, Finland.

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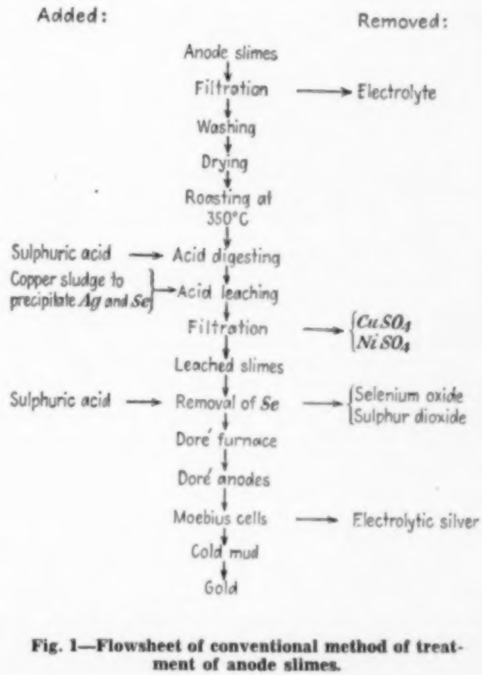

Fig. 1—Flowsheet of conventional method of treatment of anode slimes.

such a way that the precious metals could be separated in the first step in the form of high grade concentrate and with 99 pct or better extraction, then the entire treatment flowsheet could possibly be simplified and the further refinement of the precious metals concentrate reduced to bare essentials.

A "mineralogical" study of the anode slimes suggests that both gold and silver are in the form of selenides. A small amount of gold may exist as telluride. All particles are extremely fine but very strongly flocculated. The color is pitch black. This gold and silver-bearing part amounts to 15 to 25 pct of the slimes. One might say that this fraction corresponds to the sulphide minerals of a sulphide ore.

The second distinct fraction of slimes is the extremely fine, silky and brown colored nickel oxide amounting to 45 to 60 pct and representing oxide minerals of a sulphide ore.

The third fraction is metallic copper, which amounts to 5 to 15 pct and exists in coarser particles. Some of the copper may be in the form of sulphide and/or selenide.

The fourth fraction is the "gangue" of the slimes consisting mainly of quartz particles with some silicates and amounting to 15 to 35 pct.

Table I. Results of a Flotation Experiment

	Weight,		Assay, Pct			Distribution		
	G	Pct	Au	Ag	Ni	Au	Ag	Ni
Final concentrate	48	16.7	2.04	40.03	12.10	99.96	99.41	5.1
Recleaner tailing	4	1.4	0.007	0.18	53.78	0.03	0.04	1.9
Cleaner tailing	14	4.9	0.001	0.13	59.04	0.01	0.09	7.3
Final tailing	221	77.0	Trace	0.04	44.34	Trace	0.46	85.7
Head	287	100.0	0.335	6.71	39.81	100.00	100.00	100.0

Reagents Added per 2000 cc Pulp Volume	Rougher Flotation	Cleaner Flotation	Recleaner Flotation
H_2SO_4 conc.,cc	400	200	200
Reagent 208, mg	200	30	50
B-23, drops	2	1	1
Conditioning time, min	15	10	10
Flotation time, min	3	3	3

A comparison of anode slimes with a sulphide-oxide-silicate ore suggests similarity of the methods of treatment. The obvious flotation flowsheet is shown in fig. 2.

A number of experiments were run in a Fagergren flotation cell. The following observations were made:

1. A remarkable extraction of precious metals into the flotation concentrate was obtained as indicated in table I. Even after two steps of cleaning, the recovery was well over 99 pct for both gold and silver.

2. The refinery electrolyte is a good flotation medium. It is highly acid containing 200 g per liter free sulphuric acid. Selectivity of operation can be still improved by addition of fresh sulphuric acid. If, however, the primary electrolyte is removed prior to flotation by filtration and washing, the selectivity of separation in a neutral pulp is lost.

In the test shown in table I, substantial amounts of fresh sulphuric acid were used. In continuous practice the amount of acid needed could be reduced. Our experience indicates that a reduction of acid from the given figure of 400 cc per 2000 cc pulp volume to 10 cc resulted in the same grade of final concentrate and in extractions of 99.1 and 99.3 pct for gold and silver, respectively.

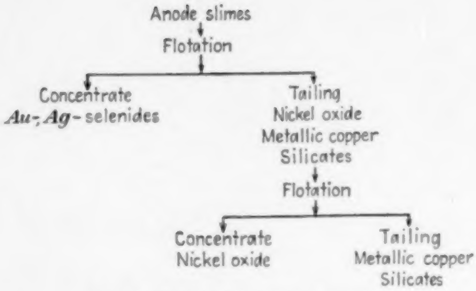

Fig. 2—Flowsheet of anode slimes.

3. The best collector found for precious metals so far has been American Cyanamid Company's Reagent 208. All tests with xanthates have been inferior due to the high acidity of the pulp. DuPont B-23 was used as frother.

4. The physical appearance of separation, the ease of its performance, and the sharpness of its end point are most unusual and seldom, if ever, enjoyed in the flotation of common ore minerals.

5. The only disappointment with the flotation concentrate has been its high nickel content. From 5 to 10 pct of the original nickel was carried over with the concentrate. Repeated cleanings reduced the amount of nickel only slightly.

6. The separation of nickel oxide from the tailing of gold and silver float was carried out with conventional reagents for oxide minerals. Before flotation the electrolyte was removed by filtration and washing. As a collector, oleic acid or tall oils may be used. A concentrate assaying 67.4 pct Ni and representing 80 pct extraction of the head was obtained in a preliminary test.

7. The presence in excessive quantities of certain organic chemicals such as glue, oils, etc. commonly used in connection with electrolysis might in adverse cases partly or totally prevent the separation of precious metals by flotation.

The economical value of flotation in the treatment of anode slimes not by any means limited to copper depends on each individual case. If the machinery and satisfactory practice for a conventional method already exist, as is the case with Outokumpu Co., the application of flotation would result in a substantial reduction of the bulk of the material and probably the simplification and/or elimination of some of the steps in the present method of treatment. The financial returns may or may not be of great interest. If, however, a company is thinking of an entirely new electrolytic plant or if the volume of operation and/or existing difficulties warrant major changes, then a preliminary treatment of the slimes by flotation might prove to be also an economical success.

Acknowledgment

This investigation was carried out on the initiative of the Outokumpu Co., especially on that of J. Kinnunen, chief chemist of Outokumpu Co. in Pori. All assaying was done in his laboratory. Thanks are also due to Mr. Eero Mäkinen, President of Outokumpu Co., for permission to publish the results obtained.

Progess Report on Grinding at Tennessee Copper Company

by
J. F. Myers
and
F. M. Lewis

This second progress report of grinding presents comments regarding ball consumption and data pertaining to the hydroscillator, which is closed circuited with the tricone mill. A study and postulate of how balls function is presented. The paper summarizes cost reduction obtained by the method of "grinding for the tail race" rather than for "mesh tons."

AT the Regional meeting in Columbus, Ohio, in September 1949, the authors presented a progress report of the first year's operation with a Hardinge tricone mill in closed circuit with a Dorr hydroscillator. The present report covers our findings on this grinding circuit to January, 1950.

In order to clarify our position, the authors wish to state that no invention or discovery is claimed. The Tennessee grinding circuit is simply an engineered arrangement of known equipment arranged to incorporate all of the recognized efficiency factors of comminution as they are known today.

As we concluded our first report, it was not clear how effective the Tennessee mode of operation would be on harder ores. The only information we had at that time was that the Bond grindability test at 48-mesh gave 6.04 g of undersize per revolution. A later test on the same sample at 100-mesh reduced this to 2.04 g, which is definitely in the harder ore class, and which makes the reported 8.80 kw-hr per ton of —200-mesh produced a very creditable figure.

Hydroscillator Operation

The machine as now developed has proved dependable and easy to operate. It starts easily under full load and responds quickly to operating adjustments. The hydraulic holes in the oscillating plate do not plug when shut down under full load. Absolutely clean hydraulic water is essential. An Elliott water screen or similar device should be an essential part of the machine to keep all scale and trash of all kinds from the hydraulic compartment. The shift operators have expressed approval of the machine.

In table I we show a typical screen analysis of the hydroscillator rake sand as compared to conventional classifier rake sand on our ore.

We have previously reported that of the 28.7 pct reduction in power, the hydroscillator contributed 6 pct, on a —200-mesh basis. This is a conservative figure. Some of our test runs indicated that the saving was as high as 10 pct. Laboratory studies by the Dorr Co. engineers indicate that still greater efficiency is possible. Nevertheless, the 10-ft plate is overflowing 2100 tons per day at 3 to 4 pct +65-mesh, which is a very creditable performance by

J. F. MYERS and F. M. LEWIS, Members AIME, are Superintendent of Concentration, and Assistant Mill Superintendent, respectively, Tennessee Copper Co., Copperhill, Tenn.

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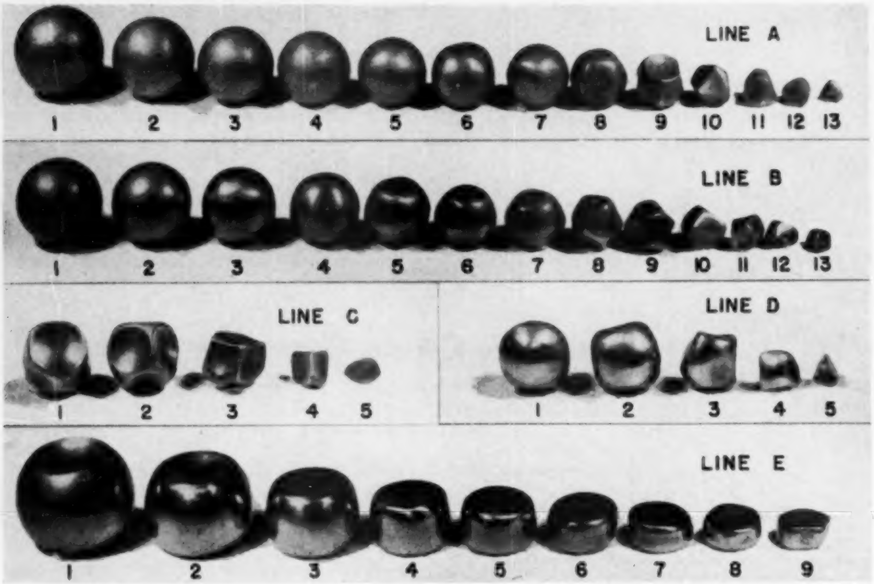

Fig. 1—Ball strings from grinding mills.

any standard. There still remains a great deal of test work to be carried out, and we do not feel that we understand the full possibilities of the machine.

Grinding Ball Wear

In our original study of savings to be gained by this unit, we estimated a 27 pct reduction in ball consumption. This was indicated by the slow-speed, ball mill operation at Hayden and other factors. After several months of operation, this saving has not developed, and the ball consumption for the first 16 months of operation stands at 1.06 lb per ton of ore, as compared with 1.05 lb per ton in the conventional mills. In both cases we were using Sheffield Moly-Cop balls, 1 in. diam.

The only explanation that we have at this time is that in the slow-speed tricone we have 34 pct more ball exposure to corrosion than in the small, fast mills. This additional corrosion must offset the 27 pct saving that we should have enjoyed.

We, as many others, have recognized the corrosion factor in all beneficiation apparatus and have ignored it as have others. Very little, if anything, is known about the relative importance of abrasion and corrosion in grinding mills.

We have always thought that the high pH value of our ball mill discharge (pH 8.5) was due to ferrous hydroxide generated by corrosion of the grinding balls and liners. It is likely that corrosion takes place to a greater degree on a heavy sulphide ore than on one with small sulphide content.

Table I. Typical Screen Analysis of Rake Sands

Mesh	Conventional Classifier	Hydroscillator
+ 48	23.4	33.6
+ 65	39.0	52.7
+ 100	60.2	75.1
+ 150	79.4	92.7
+ 200	91.0	96.9
—200	9.0	3.1

By inquiring, we learned that in some other industries a corrective method in the form of a counter electromotive force has been employed successfully to counteract corrosion. It is known as cathodic protection. We find that there is a 0.26 v drop in potential between our mill pulp and the mill shell. We are actively following up this lead with the help of several friends who have interested themselves in our corrosion problem.

During the past few months we have operated the tricone with a relatively thin discharge dilution, 50 to 60 pct solids. With a conventional speed mill such low dilutions would entail great ball and liner wear. The effective classification in the mill pool keeps a dense bed of pulp in the ball mass under all conditions, as was explained in our first report, and therefore ball wear is apparently not affected by low pulp density.

Grinding Ball Action

We considered the ball action of the slow-speed tricone as compared to the action of our conventional fast mills where cascading is prevalent. We came to the conclusion that the action is a combination of two principles. First, round balls rolling against round balls and the shell, as advocated by several authorities. Second, a "mortar and pestle" action of round balls against the concave faces of the smaller balls that have become polyhedrons in the ball paths inside the mass.

In fig. 1 we present a picture of ball strings from

our grinding mills. String "A" is from the slow tricone mill and string "B" is from the conventional 6 x 12 mill.

The wear action seems to be as follows in the tricone. The balls start out round. At approximately ⅝ of an inch the balls start to lose their round shape and begin to form polyhedrons, see ball 6, line *A*, fig. 1. As such, they cease to roll and remain in a fixed position in relation to the larger rolling balls adjacent to them. This permits the rolling balls to grind a concave surface into the smaller ball, forming a definite mortar for the pestle action of the round rolling balls. This concave face can clearly be seen in ball 9, line *A*, fig. 1.

The action is identical in the fast mills, but since there is cascading action against the breast liners and great turbulence of the balls in the outer zones, the round balls start to lose their shape much sooner, at approximately 13/16 of an inch, see ball 4, line *B*, fig. 1. The definite concave surfaces do not develop on the polyhedrons until much later in the ball life, see ball 11, line *B*, fig. 1.

In line *C* of fig. 1 is shown the characteristic progressive wear, shape and development of the concave face, as the polyhedron gets smaller in the tricone. Ball 1, in line *C*, shows one single well-defined concave surface in this size ball. Ball 2 has developed two concave faces. Ball 3 clearly shows three concave faces. The last two balls, although very small, clearly show three concave surfaces, indicating that throughout their polyhedron life they have acted as a mortar for revolving round balls.

In line *D* of fig. 1, a top view of the polyhedron of line C is shown. Note that all five balls have round top surfaces. As one watches the smooth rolling action of the tricone ball mass, the complete lack of cascading balls is obvious and, hence, there is no turbulence in the ball mass. It is, therefore, conceivable that under the first few layers of balls each rolling round ball gets itself a nonrolling polyhedron to work on. Either it or some other round ball stays in attendance of the concave surface as it goes round and round performing the mortar and pestle action. There can be no other explanation to account for the sharp, well-defined edges of the concave surfaces as illustrated in line *C*, of fig. 1. Were the polyhedrons tumbled around indiscriminately, the edges of the concaves would be rounded off and would not develop.

It is well known that ball slip takes place within the ball mass. The greatest ball movement is next to the shell, and at the center the ball movement is nearly zero. It is, of course, this slip that gives the rolling action to each round ball against its polyhedron. It is also this slip that maintains the rounded top surface of the polyhedron that has developed the concave surfaces on the bottom.

We say top and bottom of the polyhedron, which gives a mental picture of the round ball pushing the polyhedron ahead of it, as is illustrated by fig. 2*a*. It could just as well be argued that the polyhedron pushes the round ball as is illustrated by fig. 2*b*. This would account for the rounded surface on the other side of the concaves, just as well. Some few polyhedrons have two concave surfaces which seem to indicate they have a relationship with the round balls as shown in fig. 2*c*.

This poses a nice question. Of the two forces working in the mill (1) round ball against round

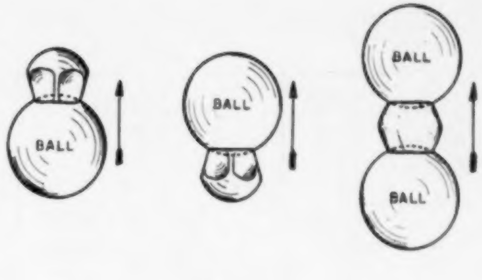

Fig. 2—Diagramatic relationship of round balls and polyhedrons.

ball and (2) the round ball and polyhedron working as a mortar and pestle, which is the more effective? Can the mortar and pestle action handle as large a particle of ore as two rolling balls? The mortar and pestle, as such, is a sliming device of the first order. Could it be that it is this effect that causes overgrinding in some of our mills today, or is it that larger particles of ore in the concaves prevent the pestle action from overgrinding the fines pocketed therein?

Within the range of ore particle size being ground (—14-mesh) in the slow tricone mill and conventional 6x12 mill, there is no question of the higher efficiency of the slow tricone. Is it because the balls stay round longer in the tricone or is it because the mortar and pestle action starts sooner on the polyhedron and lasts longer? It would be interesting to know. We thought it might shed some light on the subject to start out the make-up balls with a concave surface.

The Allis Chalmers Co. has for many years manufactured a ball called "concavex." This ball has two concave surfaces, see ball 1, line *E*, fig. 1. While it has been on the market a number of years, it has never become popular in wet grinding. The mortar and pestle idea was the thought back of making the balls this way.

As of September 27, 1949 we started adding 1¼ in. concavex balls to the tricone. Our idea was that a make-up charge of half 1 in. round balls and half 1¼ in. concavex might contribute to the grinding efficiency. By late January, after four months of observation, it became evident that our grinding efficiency was falling off and the test was dropped. Line *E*, of fig. 1, clearly indicates the reason for this. The concavex balls quickly start to get flat and thus slide in the ball mass without maintaining concave faces or staying round, and they do some rolling, as will be noted in line *E*, of fig. 1. They continue to wear down into flats, 1/16 in. thick (not shown).

In our opinion, the test did not prove or disprove anything about the mortar and pestle action.

It is with regret that we are unable to contribute some thought on this controversial subject. Cascading and impacting of balls is, of course, essential on coarse ore particles. The only thing that the tricone proves is that cascading and impacting is un-

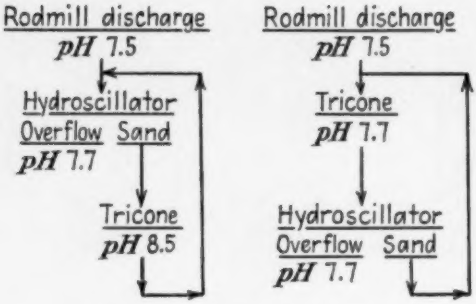

Fig. 3—pH values in grinding circuit.

necessary and inefficient on fine ore particles such as are produced by a fine crushing rod mill.

Grinding Ball Segregation

It will be recalled that the Hayden operators of the Kennecott Copper Co. reported a mass of small worn balls in the center of the ball mass of their slow speed, 7-ft mills, using 2-in. cast-iron balls[1] which they termed "the kidney." Numerous technicians, including ourselves, have felt that this was to be characteristic of slow speed mills as a whole. The slow-speed tricone, with 1 in., smooth, forged balls does not develop a kidney. Round balls of all sizes down to ⅝ in. are to be found in all parts of the ball mass, and all polyhedrons to the smallest size show the characteristic concave faces and appear in all parts of the ball mass, although to a somewhat less extent on the outside.

Metallurgical Studies

For the four-month period covered by this report, we abandoned the operation of the tricone and hydroscillator on an efficiency basis for mesh tons, and devoted our efforts to operating the unit to lower the reagent consumption and to produce the best metallurgical results.

From the 2100 tons of daily feed, we float 1300 tons of sulphide mineral as concentrate. Satisfying this sulphide surface with reagents entails considerable cost.

The main feature of reagent saving is to reduce the xanthate consumption in the bulk float. This gives a direct cost saving, of course. When the xanthate-coated particles from the bulk float are transferred to the copper mineral separating cells, a pH of 11.0 is essential. At this pH the xanthate coating on the enormous amount of iron sulphide surface is removed into the liquor and reacts with copper sulphate in the ensuing zinc separation. Less xanthate in the bulk float promotes a cost saving of copper sulphate. We agree wholeheartedly with L. E. Djingheuzian[2] that "any preparatory machine should be spoken of as a conditioner and if grinding is essential then grinding units become important conditioners."

Table II. Cost Reduction

	Old Units	New Units	Saving Per Ton	Saving Pct
Power, kw-hr per ton				
Primary and secondary	5.98	4.81	1.17	19.5
Xanthate per Ton	0.38	0.27	0.11	28.9
Copper Sulphate per Ton	0.55	0.39	0.16	29.1
Flotation Time				33.0

So far in our investigation, the best grinding efficiency on a —200-mesh basis has been obtained by discharging the open circuit rod mill to the feed well of the hydroscillator, and only the hydroscillator sand feeds the tricone.

However, the best metallurgical results are obtained by feeding the rod mill discharge direct to the tricone, fines and all, and then treating the mill discharge in the hydroscillator. We are not in a position to explain this matter and we believe it is wrong in principle to work for a clean rake sand and then confuse the issue by pouring all of the fines generated by the crusher and rod mill into the tricone. We have long known that the quicker we can grind the ore and get it to the flotation machines, the better are the results, but we likewise know that it does not tell the whole story.

Another factor comes into the picture. In fig. 3 it will be observed that in either mode of operation the feed to the flotation has a pH of 7.7 but that the tricone grinds and conditions the pulp at a pH of 8.5 in one case and at 7.7 in the other case. The higher pH is apparently detrimental, but neither does that explain everything.

Summary

Reconciling the grinding efficiency with good metallurgy is still a problem. With so many questions still unanswered it is doubtful that we have found the optimum conditions for operating the tricone-hydroscillator circuit. Nevertheless, as of this reporting, we are now enjoying some overall gains. We are equalling the best metallurgical results of the past with the savings shown in table II.

Advantage in maintenance of one large grinding unit over a multiplicity of smaller units is one useful result.

Reduction in flotation time on our three-mineral separation involving cleaners, pumps, and other auxiliary apparatus, indicates a saving of $0.050 per ton.

Conclusion

Our experience to date seems to justify the conclusion that the operation of grinding units as a means of preparation and conditioning of ore for subsequent process work is a sound engineering principle.

References

[1] W. I. Garms and J. L. Stevens: Ball Wear and Functioning of the Ball Load in a Fine-grinding Ball Mill. *Transactions* AIME (1946) **169**, 133; *Mining Technology* (March 1946) TP 1984.

[2] L. E. Djingheuzian: *Transactions* Canadian Institute of Mining Engineering. (1949) **50**, 243-257.

Rheolaveur System of Fine Coal Cleaning

by John Griffen

This paper records over twenty years' experience with the use of the Rheolaveur system in the United States, showing its ability to meet changing conditions caused by the dirtier mine output of present-day mechanical mining methods. Data are given on size ranges handled, number and capacity of units installed, maintenance experience, and operating results on two-product and three-product separations.

It is not the purpose of this paper to discuss the principles employed in the Rheolaveur system of fine-coal cleaning as these have been fully covered in the technical literature of the Institute.[1,2] Rather, our purpose will be to record the highlights of over 20 years' experience in the United States with Rheolaveur fine-coal launders, which will indicate their capabilities and costs of cleaning and their adaptability to meet the changing conditions caused by present-day mechanical mining methods.

Rheolaveur fine-coal launders are used in the United States to treat a wide variety of size ranges of fine coal, the coarsest being ½ in. to 0 and the finest about ⅛ in. to 0. In Europe feeds as coarse as ⅝ in. and as small as ½ mm (28-mesh) to 0 are being cleaned. The size ranges usually handled in this country are ⅜ to ¼-in. round to 0.

Thirty five units with a combined hourly feed capacity of 3200 tph have been installed in the United States. Individual units are cleaning as little as 25 tph, while others are cleaning as much as 200 tph. Rheolaveur fine-coal launders offer cleaning units of high capacity and are outstanding in requiring a minimum of building space per ton of input.

JOHN GRIFFEN, Member AIME, is Sales Engineer, The McNally Pittsburg Manufacturing Corp., Pittsburgh, Pa.

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Experience has shown that maintenance costs are low. Several installations were made in the Pittsburgh district from 1928 to 1930, and since that time operation has been largely two shifts per day. Minor repairs to liner plates and Rheo boxes have been required during the intervening years, but no major replacement of launders was required until 1947 and 1948. At one of these plants, launders operated for 17 years before they were replaced and during that period over 26,000,000 tons of —5/16-in. coal were cleaned.

The effect of the much dirtier raw coal produced by mechanical loading of the Pittsburgh seam is shown by the following tests. The data in table I summarize the performance of a Rheo fine-coal unit when cleaning hand-loaded raw coal. The —4-in. raw coal fed to the cleaning plant analyzed only 8.16 pct ash and 1.31 pct sulphur and contained 4.6 pct sink, 1.55 sp gr.

Two years later, in 1945, mechanically-loaded coal from the same mine was cleaned in the same plant. The feed, —4 in., then analyzed 20.10 pct ash and 1.53 pct sulphur and contained 20.2 pct sink, 1.55 sp gr. The performance of the Rheo fine-coal unit when cleaning this coal is given in table II.

It will be noted that the +48-mesh coal is cleaned almost as thoroughly as in table I. The large amount of high-ash slimes produced from the dirtier feed are responsible for the higher ash of —48-mesh washed coal in table II.

In this plant the Rheo fine-coal unit did not get the full load of refuse resulting from mechanical

Table I. Performance of Rheo Fine-Coal Unit When Cleaning Hand-loaded Raw Coal

Product	Size	Weight, Pct	Ash, Pct	Sulphur, Pct	Sink 1.55, Pct
Feed	5/16 in. +48 Mesh	86.7	8.90	1.42	8.82
	—48 Mesh	13.3	14.21	2.34	
	Total	100.0	9.61	1.54	
Washed Coal	5/16 in. +48 Mesh	79.9	5.45	1.12	1.44
	—48 Mesh	20.1	9.33	1.48	
	Total	100.0	6.23	1.19	
Refuse	Total	100.0	65.86	6.84	90.30

Table II. Performance of Rheo Fine-Coal Unit When Cleaning Mechanically-loaded Coal

Product	Size	Weight, Pct	Ash, Pct	Sulphur, Pct	Sink 1.55
Feed	5/16 in. +48 Mesh	86.8	11.45	1.83	10.42
	—48 Mesh	13.2	25.48	2.08	
	Total	100.0	13.30	1.86	
Washed Coal	5/16 in. +48 Mesh	85.0	5.58	1.28	1.78
	—48 Mesh	15.0	17.19	1.06	
	Total	100.0	7.34	1.34	
Refuse	Total	100.0	63.58	3.51	89.80

loading since the Rheolaveur coarse coal unit had already removed over 60 pct of the sink in the —5/16-in. feed coal, and only the balance reached the Rheo fine-coal unit and is reported in table II.

Rheolaveur fine-coal units can readily be operated to produce two coal products of different qualities as well as a refuse. The first grade coal can be of unusual cleanliness for special uses, while the second grade coal product consists largely of middlings for sale as steam coal or for use as mine power plant fuel. In the former case the second grade coal can be made of medium ash content as desired by a customer, while in the second case a higher ash product can usually be utilized.

A recent installation in West Virginia illustrates the possibilities of making a very high grade coking coal as the first grade product, while the second grade coal is a 12.5 to 14.0 pct ash product sold to a nearby electric power station. The coal being cleaned is a ½ in. sq to 0 size from the No. 2 Gas seam, amounting to 120 to 160 tph which is cleaned in two Rheo fine-coal units operating in parallel. Each unit is four launders high, and the overflow products of the top and next lower launders constitute coking coal; the overflow of the third launder is regulating material which is returned to the units as part of the feed, and the overflow of the fourth or bottom launder is the second grade or steam coal. The refuse is discharged by Rheo boxes on the bottom launder.

A float-and-sink test on a typical sample of raw coal is given in table III, and data on the quality of the coking coal, steam coal, and refuse are given in table IV. An examination of the data in these tables shows that the coking coal has an ash content lower than that of the raw coal, floating at 1.35 sp gr. The steam coal undoubtedly contains considerable float at 1.35 sp gr. This is required if the steam coal ash

Table III. Float-and-Sink Data, ½ in. sq to 0 Rheo Fine-Coal Unit Feed, No. 2 Gas Seam

			Cumulative			
			Float		Sink	
Specific Gravity	Weight, Pct	Ash, Pct	Weight, Pct	Ash, Pct	Weight, Pct	Ash, Pct
Float 1.35	82.6	5.80	82.6	5.80	17.4	48.50
Sink 1.35; Float 1.40	3.0	13.23	85.6	6.06	14.4	55.85
Sink 1.40; Float 1.45	2.4	19.90	88.0	6.44	12.0	63.64
Sink 1.45; Float 1.50	1.5	24.92	89.5	6.75	10.5	68.48
Sink 1.50; Float 1.60	1.3	30.06	90.8	7.08	9.2	73.91
Sink 1.60	9.2	73.91				
Total	100.0	13.23				

Table IV. Typical Operating Results with Rheolaveur Fine-Coal Units Cleaning ½ in. to 0, No. 2 Gas Seam Coal

	Ash, Pct		Refuse
1948	Coking Coal	Steam Coal	Pct Weight, Float 1.45 sp gr
June	5.27	13.50	6.4
July	5.35	13.75	7.5
August	5.36	13.22	4.1
September	5.41	14.05	4.7
October	5.79	13.85	4.6
November	5.67	13.39	7.3
December	5.55	12.59	6.5
January	5.58	12.58	7.0
February	5.78	13.57	7.2
March	5.10	14.07	5.8
April	4.98	14.05	4.3
Avg	5.44	13.51	5.95
Max	5.79	14.07	7.5
Min	4.98	12.58	4.1

Table V. Rheo Three-Product Cleaning, 12 to 0.5 mm

	Weight, Pct	Ash, Pct
Feed	100.0	12.59
Washed Coal	83.3	2.96
Middling	4.9	37.66
Refuse	11.8	70.14

content is to be kept between 12.5 and 14.0 pct because an examination of table III shows that the 1.35 to 1.45 sp gr middlings analyze 16.2 pct ash and the 1.35 to 1.50 sp gr middlings analyze 18.1 pct ash. A typical refuse analyzed 69.55 pct ash.

It is not a usual practice in this country to produce a high-ash middling or second-grade coal for mine power plant fuel, but this is quite common practice in western Europe. I have seen the records of a Rheolaveur washery in the Ruhr, Germany, where middlings of 35 to 40 pct ash were made. The 12 mm (½ in.) to 0 coal was cleaned for a coke plant. Raw coal of this size was first screened at 3.5 mm (⅛ in.) and dedusted at ½ mm (1/50 in.) and about 60 pct of the 3.5 to 0.5 mm was cleaned on dry tables, the balance of this size with the 12 to 3.5 mm, and the middlings from the dry tables were cleaned in a Rheolaveur fine-coal unit. Typical results are given in table V.

This Rheolaveur fine-coal unit was cleaning 140 to 150 metric tph of input.

References

[1] J. T. Crawford, C. P. Proctor, and J. A. Jounkins: Launder and Table Washing of Fine Coal. *Trans.* AIME (1940) **139**, 269.

[2] Coal Preparation. (1943) New York, AIME, and revised edition (1950).

Special Methods for the Beneficiation of Glass Sand

by Paul M. Tyler

Higher freight rates and better methods of beneficiation now may make it more economical to open inferior deposits closer to a glass factory than to work higher-grade deposits farther away. Nature of impurities and special treatments as well as common practice of sulphuric acid leaching are described.

HISTORICAL concepts of the economics of the glass-sand industry are changing rapidly. The greatly expanded demand for glass containers combined with higher freight rates on raw materials and manufactured products have induced a migration of glass factories toward densely populated centers and the creation of new standards of place value for substandard sand deposits. This migration has been facilitated by the construction of pipelines to bring cheaper natural gas and liquid fuel to large cities and may be further speeded by the adoption of modern mineral dressing methods to permit economical utilization of local raw materials.

Our national resources of naturally high-grade silica sand are abundant, but most of the best deposits are situated so far from large centers of population that it now costs \$3.00 to \$5.00 a ton to carry the sand from the mine to the optimum site for a factory making beverage bottles, scientific and electrical glassware, and miscellaneous blown and pressed ware for local markets. It follows that on the basis of freight savings alone it may be economical to use material from inferior deposits closer to the factory even though the f.o.b. cost of mining and treatment may be much higher than at a higher grade deposit farther away.

Particle-size distribution is a feature of glass-sand specifications. As long as glassmakers insisted upon having their sand principally coarser than 100-mesh, any material was automatically excluded from consideration that could not be purified without fine grinding or other drastic treatment which greatly reduces the size of the quartz grains. However, recent large-scale tests in Norway[1] showed that finely-crushed quartz used in a soda-lime-silica glass batch caused it to melt and become refined appreciably faster than when ordinary coarse (Belgian) sand was employed. Dust losses were negligible. Since the pre-war cost of sand was only 3 pct of the manufacturing cost of the glass, it was deemed economical to pay twice as much for crushed quartz because of the saving in wear and tear on the furnace walls. In the author's unverified opinion, sand ranging in size between 150 and 400-mesh would probably be superior to that in the more usual range of 20 to 100-mesh. Former objections to fine sand (other than dust) were doubtless fostered by the natural concentration of impurities in the finer sizes.

Ordinarily the most objectionable, as well as the commonest, impurity from the standpoint of the manufacturer of any but the cheaper qualities of colored glass is iron. No natural sands are really pure silica. Even water-clear quartz crystals are likely to contain impurities in the form of solid solution as well as inclusions which cannot be eliminated except by chemical treatment that will break down the silica lattice. Pure white pegmatite quartz will usually analyze 0.01 pct or more of iron oxide, and ordinary vein quartz and the quartz grains in most

PAUL M. TYLER, Member AIME, is Consulting Mineral Technologist and Economist, Bethesda, Md.

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crystalline rocks may be quite impure. Commercial glass sand is almost always won from sedimentary deposits, including unconsolidated beds and lightly-cemented, friable sandstones. Such deposits usually contain grains of other minerals besides quartz, and during and after sedimentation they may be invaded by surface or underground waters, which may precipitate impurities upon the surfaces of the sand grains. It is true that organic acids, such as occur in peaty soils, may reduce ferric iron to form relatively soluble ferrous iron compounds, which are leached out and finally eliminated. As a rule, however, the iron content of a deposit is substantially higher than that of its constituent quartz grains.

The Mineral Dressing Problem

Given a representative sample of a deposit, the first step in determining its amenability to commercial methods of purification is a petrographic study. Ordinarily, this calls for a screen analysis, each size fraction then being examined by microscopic grain counts, supplemented when necessary by heavy-liquid, high-intensity magnet, and other tests. The object of this preliminary examination is to identify all the minerals and to determine whether the iron occurs principally or significantly in one or more of the following ways:

1. In clay nodules, ferruginous clay bond, or kaolinized feldspar soft enough to be dispersed in water and eliminated by light scrubbing and water washing.

2. As adherent crusts or stains on the surfaces of the quartz grains.

3. As disseminated or penetrated particles in the quartz grains.

4. As discrete particles of limonite, magnetite, ilmenite, pyrite or other heavy iron minerals.

5. As discrete particles of minerals containing iron as an accessory constituent, such as hornblende, garnet, glauconite, or biotite.

6. As an impurity (solid solution?) in discrete particles of muscovite, sericite, rutile or zircon.

7. As adherent iron-oxide crusts or stains which alter (activate) the surface characteristics of heavy mineral grains.

8. Mechanically included in some lighter mineral such as cracked or weathered feldspar in arkosic sands.

Methods for removing soft clay are too well known to require discussion in this paper. They are common to ordinary sand and gravel preparation and correspond to desliming of ore pulps. Mining and treatment methods and costs at a small California plant have been described in detail elsewhere[2] and numerous examples of current practice are covered in various trade journal articles of which only a few are listed.[3,4,5] In a mineral dressing laboratory, a simple blunging test is conveniently made in a Fagergren or other type of flotation cell with a 1 : 1 ratio of sand to water. The agitation can be done stepwise with or without additions of NaOH, Calgon, or other dispersing agents. More drastic treatment is by impact grinding in a pebble mill followed by washing.

Discrete particles of heavy minerals are almost invariably smaller than the average quartz particles and thus are almost ideally prepared for easy removal by gravity separation. Tabling is the time-honored method but the Humphreys spiral and other devices should also be tried. Magnetic separation is sometimes indicated. High-intensity magnetics will ordinarily remove grains that contain as much as 1.5 pct Fe_2O_3.[6] Electrostatic separation, according to the author's experience, is likely to be considered merely as an auxiliary process to separate zircon, rutile, or other commercial minerals from a mixed heavy mineral concentrate, but this situation may change.

The characteristically small size of the heavy mineral particles is also a favorable factor in froth flotation. Flotation likewise may be effective in removing locked, iron-encrusted, or heavily stained particles of quartz and other minerals regardless of their apparent specific gravity. Petrographic examination may indicate whether an iron stain that penetrates deeply into the surface of the sand grains actually covers enough of the surface to activate it for flotation. Anionic collectors (fatty acids) are commonly used for removing iron-bearing impurities, whereas cationic collectors may be more effective for removing feldspar, mica, or sericite. The use of a sulphur-phosphorus derivative of cresol (Aerofloat No. 15) along with pine oil, preferably in a heated pulp, has been patented by Haddan[7] as a method of removing iron impurities. Mica sometimes can be removed in settling boxes or troughs. The Humphreys spiral has been employed experimentally to recover a fine flake mica product from mill tailings[8] but it is not known whether such treatment would scalp off enough mica to clean glass sand properly.

Although the treatment of glass sands by any of the common mineral dressing methods may involve certain minor peculiarities, the general principles thereof are stated in standard textbooks and are quite familiar to all mineral dressing engineers. In this paper, therefore, it has seemed desirable merely to call attention to the importance of possible activation of various nonmetallic mineral surfaces by iron compounds or ions and the complications caused by clay or slime coatings on the proper filming of such surfaces for froth flotation.

Before turning to the consideration of special methods of glass-sand treatment, however, it may be noted that a brief outline of suitable experimental procedures for laboratory tests may be found in a bulletin of the Rutgers University Bureau of Mineral Research.[9] The same reference summarizes available estimates as to typical costs of treatment by specified processes as listed below. These estimates are geared to conditions during and immediately after World War II and, of course, are subject to wide variation according to the material treated, the size of the operation, and the desired quality of product.

Table I. Approximate Costs of Beneficiation

Method	Cost per Ton of Sand
Paddle, rake, or screw washers and settling cones	$0.05 to $0.15, depending on method and grades desired
Mining and washing unconsolidated sand	About $0.35
Tabling	$0.10 to $0.40
Froth flotation	$0.10 to $0.75, depending mainly on cost of reagents
Elutriation	No data
Acid leaching	About $0.75
Magnetic separation	$0.10 to $0.18 (ilmenite removal)
Electrostatic separation	Upkeep low; current requirement may average 4 kw per hr

Approximate costs of beneficiation are given in table I.

Table II. Preliminary Tests Without a Reducing Agent

	Pct
25 pct HF stirred 5 min	0.0075 Fe_2O_3
15 pct HF stirred at intervals over 3½ hr	0.012 Fe_2O_3
10 pct HF stirred at intervals over 3½ hr	0.012 Fe_2O_3
5 pct HF undisturbed overnight	0.014 Fe_2O_3
2 pct HF undisturbed for 48 hr	0.027 Fe_2O_3

Chemical Treatments

Acid leaching has been used with technical success with mineral acids alone or in combination with various reducing agents. It has generally been considered expensive, but under the conditions outlined at the beginning of this paper, it may not be too expensive where cheaper methods fail. Although this method may be effective in dissolving small amounts of finely-divided discrete particles of iron and titanium minerals, its principal use is for removing limonite specks and stains. When clay or other impurities are present, preliminary scrubbing and desliming are usually indicated.

Processes using many different combinations of chemicals have been patented and some of these processes have been employed in commercial plants. Among the latter is the Adams process[10] which has been used in England and which was designed to avoid prolonged heating and the many difficulties surrounding the use of strong acids. It employs a water solution containing from 0.25 to 2.0 pct of sodium acid oxalate (tetroxalate) by weight and about one fifth that quantity of ferrous sulphate at a temperature of about 180°F. Sand, which after thorough washing showed an average of 0.07 pct Fe_2O_3, gave an analysis of 0.035 pct after this treatment, and selected sand was reduced in the same plant from about 0.045 to under 0.025 pct Fe_2O_3. Curtin (U. S. Patent 2,198,527, April 23, 1940) discusses countercurrent leaching with hot dilute oxalic acid, the acid being subsequently regenerated by precipitating calcium oxalate from the spent leach liquor and treating with H_2SO_4. The use of SO_2 is mentioned by Poole,[11] and Gregorjeff and Kaschirina[12] report reduction of the iron oxide content of one sand from 0.16 to 0.01 pct Fe_2O_3 and of another sand from 1.5 to 0.25 pct by digestion in a saturated solution of SO_2 in water at 20 pct solids. The use of zinc hydrosulphite is another possibility although the writer has not tried it on sand and has not noted references to such use in the literature. Mixtures of $SnCl_2$ and HCl are reported as being used successfully but would seem to be rather costly.

Sherlock,[13] experimenting with a British sand containing 0.07 pct Fe_2O_3, which remained insoluble in HCl except in high concentrations and elevated temperatures, obtained optimum results with HF and titanous chloride. The results of preliminary tests without a reducing agent are shown in table II.

The original iron content of the sand used in the above-mentioned tests was 0.045 pct. After water washing this was lowered only to 0.031 pct whereas the desired product was optical glass quality sand with less than 0.01 pct Fe_2O_3. Although the foregoing tabulation indicated considerable action by rather weak solutions, the handling of the acid was still difficult. Accordingly, a search was made for a suitable accelerator. It was found that the addition of a 1 pct solution of titanous chloride to a 1 pct solution of HF was effective in removing the coating in 5 min at ordinary temperatures. Titanous sulphate worked equally well but other reducing agents did not. In the plant built to use this patented process (British Patent 555,241, Nov. 7, 1941) the sand remained in contact with the solution for about 10 min. Neutral salts, such as sodium fluoride or sodium silico-fluoride, produced a similar effect to HF even in alkaline solution, indicating that the mechanism of the process is first to loosen the clay crust and then to reduce the iron to a soluble salt, presumably a fluo-titanate. No data are given as to consumption of chemicals. Both HF and $TiCl_3$ are rather expensive, but if they prove more effective than cheaper chemicals on certain sands, they may still be worth considering, especially since the reactions take place at atmospheric temperatures.

Sawyer[14] describes an Ohio plant for treating "glass rock" carrying 4 pct clay and 0.07 pct Fe_2O_3 on the surface of the grains. Rock from the quarry was washed and scrubbed with water so as not to break up the sand grains (which are already rather fine); conditioned with NaOH, fuel oil, fatty acid, and pine oil; and then treated in flotation cells. After flotation, the partially purified sands are dewatered, dried at 272°F, and repulped. Then H_2SO_4 is added at the rate of 25 lb per ton and the mixture run into large wooden tubs where it is heated by steam to 220°F. A higher temperature would decompose the ferrous sulphate. Finally, the sand is washed with water, dewatered, dried, and screened. Sulphuric acid leaching has also been employed in California.[15] Although the cost of the acid is rather low, this process requires acidproof equipment and usually a boiler plant to furnish steam for heating solutions.

Attrition Scrubbing and Grinding

The attrition scrubber devised by the Bureau of Mines at College Park, Maryland,[6, 16] for preparing clean surfaces for froth flotation may be used to rub off the coating on mineral grains without breaking the grains. It is essentially a drastic blunging device with a high-speed rotor and stationary baffles. One model has a vertical shaft carrying radial spokes or blades; another has vertical blades, a design inspired

by the Fagergren flotation cell. The stationary baffles are placed so that the clearance between them and the rotor is only a few times the diameter of the largest grain in the pulp treated. Although the machine may be run on dry sand, it is designed to produce maximum turbulence in a water suspension. The grains in the pulp are forced to collide and rub against one another. While this action provides drastic attrition or "scuffing," the metal surfaces are rubber-covered so as to cushion impact and thus avoid breaking the grains. The rubber, of course, also minimizes wear on the machine due to erosion. Laboratory operating data with this machine on a large number of samples are reported by Dasher and Ralston,[6] and the writer can testify as to successful employment on numerous tests. Unless the stains are exceptionally tough, they can be removed almost completely by an expenditure of 2½ to 10 hp-hr per ton.

A more conventional means of accomplishing the removal of indurated stains and compact crusts is attrition grinding in a pebble mill. Laboratory and pilot plant tests and a design for a 300-ton commercial plant using this method have recently been described by Poole.[11] Essential features of this flowsheet are preliminary grinding in a 4 x 8-ft rod mill in closed circuit with a 10-mesh screen followed by a magnetic separator and then by a thickener ahead of a 2 ft 6-in. x 7-ft pebble mill operated with ¾-in. pebbles and 60 pct solids. The sand is ground by attrition only in the pebble mill which is in closed circuit with a bowl classifier which discharges a product ranging between 150 and 400-mesh. The raw material is Falls Creek sandstone, a pale buff, soft and friable rock averaging 98.75 pct SiO_2, 0.12 pct Fe_2O_3, 0.67 pct Al_2O_3, and 0.10 pct TiO_2. The iron oxide content of the product was reduced by this treatment to 0.025 pct.

Attrition grinding is defined as a treatment which will reduce the size of sand grains by scouring or rubbing their surfaces but not by cracking or impact grinding of the grains themselves. Since the object is to have the pebbles merely slide over the surfaces of the sand grains, the pebbles cannot be too heavy. According to Poole[11] the top limit of pebble size is about ¾ in. and the optimum ratio of sand to pebble charge is 1.1 : 8. For impact grinding the latter ratio is characteristically much higher, say 3 : 8. Theorists might reason that good results might be obtained by a light pebble charge which would merely stir up the charge and favor self-grinding, but with fine sand the friction between the sand grains alone is likely to be too small to be truly effective.

The cost of the pebble-mill treatment, based on a 300-ton plant and 1943 prices, including 10-year amortization charges, is estimated at $2.00 a ton.

Summary

In conclusion, the best method of beneficiating a given glass sand depends essentially upon the nature of the material and, more particularly, upon the mode of occurrence of the iron and other impurities. The simplest treatment is desliming with or without incidental grinding or scuffing to break up and disperse clay or other semicolloidal iron-bearing material so that they can be washed out in a suitable classifier. When the impurities are present as discrete particles or fine free minerals, standard mineral dressing techniques such as gravity separation or froth flotation are likely to be effective. When the impurities occur largely in the form of a stain or crust on the surfaces of the sand grains, either chemical (leaching) or mechanical (attrition grinding) treatment is indicated. Froth flotation can remove heavily stained grains but only when most of the surface is so covered. Quartz grains containing internal specks or disseminated impurities can only occasionally be removed by gravity or magnetic separation. Iron dispersed or in solid solution in the quartz cannot be removed by any means short of destruction of the crystal lattice.

References

[1] F. Gjesmoe: Fine-Grained Sand for Glass Melting. *Journal* Soc. Glass Technology (Oct.-Dec. 1947) **31**, 259-263.

[2] E. Shaw: Mining and Milling Methods and Costs at the Glass Sand Plant of P. J. Weisel, Inc., Corona, Calif. U. S. Bur. of Mines, *I. C.* 6937 (1937).

[3] H. E. Swanson: Washing and Screening for Grade Glass Sand. Oklahoma Silica Sand Co. *Rock Products.* (Dec. 1946) **49**, 78-80.

[4] E. C. Hoeman and R. C. Redfield: Processing Industrial Sand from River Deposits. *Rock Products.* (June 1947) **50**, 105.

[5] W. B. Lenhart: Million Dollar Sand and Gravel Plant. *Rock Products.* (July 1948) **51**, 80-84.

[6] J. Dasher and O. C. Ralston: New Methods of Cleaning Glass Sands. *Bulletin* Amer. Ceramic Soc., (June 1941) **20**, 187-195.

[7] British Patent 462,213, April 6, 1936 (Abstr. *Journal* Soc. Glass Technology (1937) **21**, 236A).

[8] R. Adair, W. T. McDaniel, and W. R. Hudspeth: A New Method for Recovery of Flake Mica from Washing Plant Tailings. R.I. 1, Minerals Research Lab., Asheville, N. C. (Oct. 1949).

[9] A. S. Wilkerson and J. E. Comeforo: Some New Jersey Glass Sands. Rutgers Univ. Bur. of Minerals Research *Bulletin* No. 1 (1948).

[10] F. W. Adams: New Process for Removal of Iron from Silica Sands. *Journal* Soc. Glass Technology (1935) **19**, 118-119; Purification of Glassmaking Sands on a Commercial Scale. *Ibid.* (1937) **21**, 409-414.

[11] J. P. Poole: Glass-Sand Beneficiation Process. *Journal* Amer. Ceramic Soc. (July 1949) **32**, 237-241.

[12] P. N. Gregorjeff and W. T. Kaschirina: Removal of Iron from Clays and Sands. *Sprechsaal* (1937) **70**, 2. (Abstract *Journal* Soc. Glass Technology (1937) **21**, 84).

[13] A. Sherlock: Methods for the Purification of Sands for Glassmaking. *Journal* Soc. Glass Technology (1945) **29**, 268-272.

[14] E. C. Sawyer: Refining of Silica Sand for Glass Manufacture. Ohio State Univ. *Engineering Experimental Station News* (1947) **19**, No. 2, 57-60.

[15] Brentwood Sand. *Engineering Mining Journal* (Dec. 1937) **138**, 29-31.

[16] J. E. Norman and O. C. Ralston: Conditioning Surfaces for Froth Flotation. *Transactions* AIME (1939) **134**, 65-80.

Magnetic Fields Associated With Igneous Pipes In the Central Ozarks

by Charles R. Holmes

Fig. 1—Location map of intrusive.

MORE than 70 igneous pipes and dikes are known to occur in Cambrian sediments throughout an approximately circular area of about 75 sq miles in southwestern Ste. Genevieve County and southeastern St. Francois County, Mo., across the southern portion of the Farmington anticline, which is outlined in fig. 1. From evidence obtained from fossil fragments taken from some of the intrusives, and from the similarity in composition of the more basic intrusives to those of known age in surrounding states, these pipes and dikes are believed to be at least post-Devonian and probably Cretaceous in age. As first postulated by Rust,[1] these intrusives are believed to represent explosion tubes, or diatremes, punched through great thicknesses of solid rock by gaseous pressure.

The particular intrusive studied is located about 1 mile north of the small town of Avon at an elevation of 880 ft in the E½ SE¼ NE¼ of sec. 2, T. 35N, R. 7E. Outcrop is through the Bonneterre dolomite in two separated exposures. The larger outcrop extends east and west for about 50 ft along a tributary in a small ravine (fig. 2). The smaller exposure occurs as a few scattered patches over an area 20 ft square about 100 ft to the north. The area between the exposures is covered, but scattered patches of the Bonneterre dolomite outcrop throughout the area surrounding the diatreme. Where exposed, contact relations show the surrounding dolomite to be shattered and metamorphosed to a fine-grained rock for a distance of 15 to 20 ft from the intrusive.

The igneous rock occurs as a dark greenish-gray porphyry near the center of the diatreme and as a fine-grained, greenish-gray material containing lapilli and metamorphosed fragments of dolomite near the contact with the country rock. The most abundant original mineral of the porphyritic rock is olivine now largely serpentinized as the result of extensive hydrothermal alteration at time of emplacement. The most common constituent of the fine-grained rock, which occurs near the border of the intrusive, is calcite. Mica is a common constituent of both types of rock, occurring as tufts or flakes. Magnetite makes up about 2 pct of the diatreme and occurs as small irregular grains or streaks of tiny grains. Singewald and Milton[2] have termed the rock of this body an "augite-free alnöite."

Susceptibility measurements of pulverized specimens were made from selected samples taken from both the diatreme and the surrounding country rock. The susceptibility of the Bonneterre dolomite was found to be less than 40×10^{-6} cgs (centimeter-gram-second) units and that of the metamorphosed contact dolomite less than 200×10^{-6} cgs. Measurements of individual samples of the igneous rock showed a wide variation in susceptibility between limits of 1000×10^{-6} cgs to 8000×10^{-6} cgs, with the maximum difference occuring in samples taken only a few feet apart. The maximum value was exhibited by a highly weathered sample taken from the northern outcrop, while minimum susceptibility was determined from a partially weathered sample taken only a few feet away. These extremes are believed to represent small local concentrations of stringers and grains of magnetite crystals in the igneous rock.

A detailed magnetic survey of the horizontal and vertical components of the earth's magnetic field was made over the diatreme area. Stations were occupied at 20-ft intervals as shown by the grid on fig. 2. When greater detail was desired, intervening stations were also occupied. The field instruments were supplied by the Department of Geophysics, St. Louis University. The magnetometers are the Schmidt balance type constructed by the Ruska Instrument Co. These instruments are temperature compensated and have a standard sensitivity of 10 gamma. Calibration of the instruments

CHARLES R. HOLMES, formerly with the Department of Geophysics, St. Louis University, St. Louis, Missouri, is now associated with the New Mexico School of Mines, Socorro, New Mexico.

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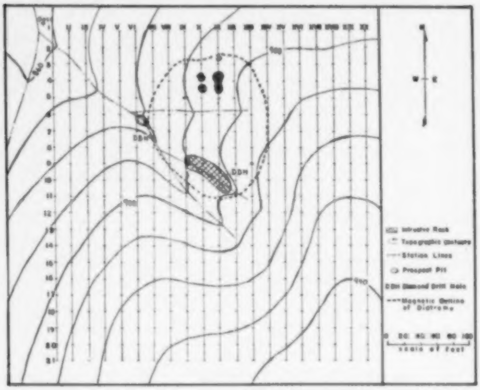

Fig. 2—Geological, topographical, and station map of intrusive.

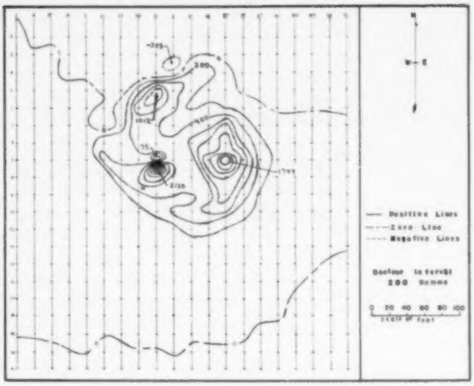

Fig. 3—Vertical isanomalic map.

was accomplished in the field by means of a Helmholtz coil, which was furnished by the manufacturer of the magnetometer. The vertical field balance was found to have a sensitivity of 9.7 gamma per scale division. Sensitivity of the horizontal instrument was determined to be 10.2 gamma per scale division. The sensitivity of each instrument was redetermined at regular intervals throughout the course of the survey and was found to remain sensibly constant. Some slight variation was noted in the second decimal place of the scale constant for each instrument, but the magnitude of change was never sufficient to alter the value in the first decimal place. In the present survey, all calibrations were made at the base station.

A self-recording base station instrument was used to supply diurnal corrections to the data obtained from the field survey of the vertical magnetic intensity. This instrument was also manufactured by Ruska and photographically records temperature changes and diurnal variations in the vertical component of the earth's magnetic field over a 15-hr interval. Timing marks are placed on the record at ½-hr intervals. The sensitivity of the instrument was determined to be 10.2 gamma. The records made by the instrument were supplemented by an additional diurnal curve, which was obtained each day from readings taken every 2 hr at the base station with field instruments. Average diurnal variation was less than 50 gamma and usually less than 20 gamma for any single 2-hr period.

The magnitude of the change in the anomalous magnetic field over the diatreme necessitated the use of auxiliary magnets with the field instruments. The moments of these magnets are subject to variation with change in temperature and thus introduce a small unavoidable error into the survey. The level bubbles of these particular instruments are extremely sensitive to the effect of the sun's rays, and, although care was exercised in orienting and leveling the instruments, some small additional error may have resulted. Because of these factors, base lines were run for the survey to provide checks and corrections to the other lines. Base line corrections are applied linearly. Particular care was exercised in taking readings along the base lines. They were run during cool cloudy periods, when disturbing influences were at a minimum. From the base line readings and from numerous other checks made during the course of the survey, it is believed that the overall accuracy of the magnetic survey after corrections is to within 10 gamma.

Fig. 3 represents an isanomalic map constructed for the magnetic anomaly in the vertical intensity over the body. Three separate positive areas are present within the limits of the anomaly. The most intense high is centered about 10 ft south of station 9 on traverse line IX. This high has a range of over 2100 gamma but is limited in areal extent. Another positive area occurs just to the west of the smaller igneous exposure at station 5, line IX, with a maximum anomaly of 1012 gamma. A much broader positive area occurs centered over station 9, line XIII, northeast of the main outcrop. The maximum value here is 1744 gamma. An inspection of the map reveals that these three "magnetic highs" are superimposed upon a much broader positive anomaly nearly circular in extent.

The horizontal component of the anomalous magnetic field over the same region is shown in fig. 4. The maximum variation in intensity again was found just south of station 9, line IX, where a low of 1820 gamma occurred. The anomaly centered over station 9, line XIII, shows a range of about 1200 gamma and that at station 5, line IX, a minimum value of 1007 gamma.

Fig. 5 represents the north-south profile for the two anomalies centered on line IX. The appearance of the profile indicates that the intensity values near station 9 are abnormal. Since at this station the horizontal profile is displaced far below what should be its normal relation to the vertical profile, with respect to a zero axis, the anomaly cannot be explained by the mere near-surface concentration of magnetic material. The larger exposure of the diatreme outcrops on the line at station 9 as a small cliff about 5 ft high and facing south. The greatest variation in both horizontal and vertical intensities occurs within 5 ft of the cliff face and most probably results from flux concentration at the edges of the outcrop. The anomaly to the north at station 4 is much broader in areal extent in relation to range of magnitude than is the disturbance at station 9. In this case, the displacement of the horizontal curve below the zero axis reflects the normal negative value observed north of the center of a disturbing body in the northern hemisphere. Although there may be other factors

contributing to the magnetic disturbance at station 4, the regular character of the magnetic intensity curves and the elliptical pattern of the lines of the isanomalic maps in the vicinity of the station suggest the concentration of magnetite grains as the basic cause of this local anomaly. Higher than average susceptibility values exhibited by rock specimens taken 10 ft south of station 3 on line X lend credence to this inference. The result of the mineral concentration is also evident in the magnetic intensity values for adjacent stations on lines VIII and X.

Although the data show that the magnetic intensity profiles for lines X through XIII exhibit a considerable diversity in range of intensity and appearance, each line possesses three distinct intensity peaks centered approximately at stations 4, 9, and 12. Blum[3] investigated the vertical magnetic field over two protrudent volcanic pipes in Colorado and observed that the intensity values over the circular edges of the pipes were greater than toward the center and that the southern half of the circumference has a stronger field than the northern half. He attributed the latter to the inclination of the earth's magnetic field. These observations find analogy to the present study with the exception of the relative magnitude of the magnetic field toward the center of the diatreme. The intensity values over this body are a direct reflection of its uneven near-surface configuration. The circular edge of the diatreme is for the most part buried beneath a covering of soil and mantle rock. Thus the point of maximum anomaly will be shifted toward the main outcrop. Other things equal, the relation of the magnitude of the magnetic intensities toward the center of the body to those over its edges will depend on the sharpness of the edge and its distance beneath the surface at the point of observation. Therefore, the edge effects serve to outline the plan shape and extent of the diatreme. The contact of the intrusive with the country rock is exposed just west of station 7, line VII, and may also be observed 10 ft north of station 3 on line XI. Although edge effects are shown by the observed magnetic data for intermediate lines across the body, there are no breaks in the curves for lines VII and XIV. A comparison of the character of these two lines, together with known contact relation near line VII, indicates that line XIV bounds the eastward extension of the body. Then if the

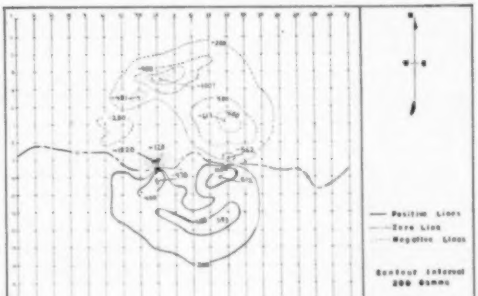

Fig. 4—Horizontal isanomalic map.

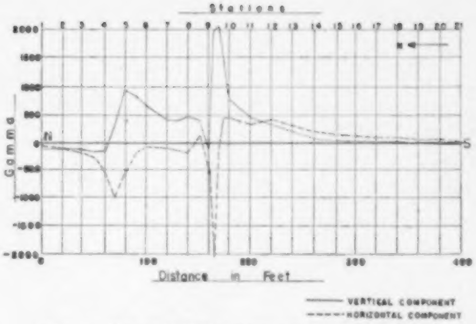

Fig. 5—Magnetic intensity profiles for line IX.

normal southward displacement of the edge effects due to the inclination of the earth's magnetic field is considered, the diatreme can be delineated as an elliptically shaped vertical pipe with the long axis of the ellipse oriented in a north-south direction. The probable outline of the intrusive is shown in fig. 2.

From the outcrop pattern of the intrusive and the sharp changes in intensities of the magnetic profiles north of the main outcrop, it was first thought that the diatreme was not continuous beneath the covered area between the two outcrops but either branched some distance beneath the surface or was separated into two distinct bodies. To test these relations and to determine a possible limiting depth to the diatreme beneath the covered area, a two-dimensional analysis of the body was made for the north-south line across the center of the diatreme. The assumptions on which the analysis is based are: (1) that the body is infinite in extent vertically downward and perpendicular to the magnetic meridian, (2) that there is no variation in susceptibility throughout the body, (3) that near surface effects of flux concentration along corners and edges can be disregarded. None of these assumptions is valid in the actual case, but if the limiting factors are considered, some conclusions still can be made on the structure of the body. For the purpose of analysis, the body was divided into three adjacent blocks based loosely on known contact and outcrop relations along line XI. The northern block was assumed to be 20 ft in width along the traverse line and to lie 5 ft below the level of observation with its northern and southern edges at points 10 ft south of station 3 and 10 ft south of station 4, respectively. The top of the southernmost block was assumed to be at the same level with its north edge at station 7 and its south edge at station 11. Then the elevation of the top of the central block was altered with respect to the other two until the resultant of the curves computed from all three blocks approached the observed curve in character. A depth of 20 ft below the level of observation was found to be the maximum depth that could be taken for the central block and still have the theoretical curves approach those constructed from the field data. Under the assumptions made, it constitutes a limiting depth to the body for the covered area between the outcrops and indicates that the diatreme cannot be taken as two separate bodies. It is readily admitted that a two-dimensional analysis is not directly applicable to the calculation of the exact configuration of a pipe-shaped body. The analysis is undertaken only to show the maximum depth to the covered section, and, as such, serves the purpose.

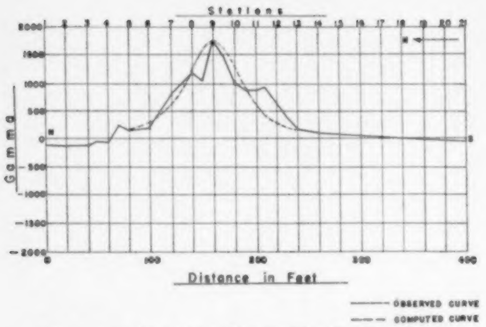

Fig. 6—Vertical intensity profiles for line XIII.

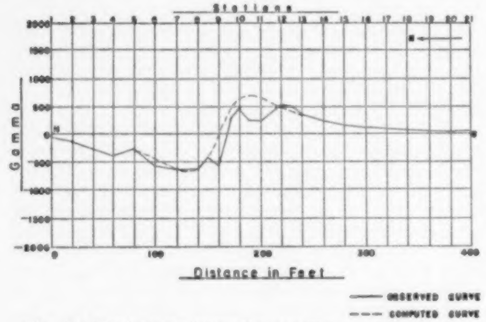

Fig. 7—Horizontal intensity profiles for line XIII.

The appearance of the isanomalic maps suggests that the anomaly centered over station 9 of line XIII results from the concentration of stringers and grains of magnetite and that the resultant magnetic field in the immediate vicinity of the station might be represented by an isolated magnetic pole. Total intensity vectors constructed from the two observed magnetic components intersect along line XIII from 40 to 50 ft beneath the surface at station 9. Figs. 6 and 7 illustrate the correspondence between the actual curves constructed from the observed data and type curves calculated from an equivalent pole at the minimum depth of 50 ft. Even with the pole located at this minimum depth, magnetic intensity values calculated from it for stations east of line XIII are considerably higher than observed values. The rapid decrease observed in the magnitude of the magnetic field east of line XIII indicates that the anomaly centered here cannot be attributed entirely to the effects of mineral concentration but must be at least partly caused by near-surface effects of the underlying body.

A supplemental gravimetric survey was also carried out over the area in an attempt to confirm the magnetic findings. The instrument used was a small Atlas gravity meter, model F, with a standard sensitivity of 0.1 milligal. Two traverses were run: one north-south along line XI and the other east-west along the line of station 7. Density determinations were made from selected rock samples by the method of weighing in water and in air. A few additional determinations were made from rock specimens cut into regular geometrical shapes. The average density of the unweathered porphyritic rock of the diatreme is 2.89. The fine-grained igneous contact rock containing dolomitic inclusions averaged 2.68 and the densities of the metamorphosed country rock just outside the contact, 2.70. Density of the unaltered Bonneterre dolomite is 2.73. The density of the weathered alnöite depends on the degree of decomposition and is extremely variable. The average value for highly weathered solid samples is 1.89. These values fix the maximum effective density contrast between the diatreme and the surrounding country rock at 0.16.

The maximum anomaly to be expected over the center of a vertical circular cylinder that is infinite in depth can be calculated from the equation:[4]

$$g = 2\pi k \zeta r$$

where r = the radius of the body in centimeters;

ζ = the density contrast;

k = the gravitational constant, which is 6.664×10^{-8} cgs

Assuming the diatreme to be circular in plan with a mean radius of 2280 cm, the anomalous value that results from a density contrast of 0.16 is only 0.162 milligals. Therefore any interpretation of the gravitational field requires that the resolution of the gravity data be within this limiting value. As conducted, the gravity survey constituted a trial run for the instrument. It since has been observed that the drift of the gravity meter is nonlinear for 3 to 7 min after the instrument is in position for the observation. The possible errors compounded from nonlinear instrumental drift and elevation corrections to the observed data reduce the limit of accuracy of the survey to 0.2 milligals. This value exceeds the magnitude of the theoretical anomaly and indicates that, except under exceptionally favorable conditions, the gravity method is of little value in the investigation of these igneous pipes.

Although the anomalous gravity field is not definitive, the other evidence is sufficient for a solution of the problem. The geologic relations indicate that the body was intruded into the surrounding sediments with the explosive violence of a diatreme. The analysis of the observed data indicates that the diatreme is an elliptically shaped pipe intruded vertically upward into the surrounding sediments with its long axis oriented in a north-south direction. The concentration of magnetic materials and the near-surface effects of flux concentration over corners and edges of the underlying field greatly influence the character and magnitude of points of maximum anomaly over the intrusion. From the edge effects in the anomalous magnetic field and the known contact relations, the causal body is believed to measure about 160x140 ft in the plan section. The inference as to the form of the diatreme is in substantial agreement with the results of a previous investigation conducted by the Missouri Geologic Survey.[5]

References

[1] G. W. Rust: Preliminary Notes on Explosive Volcanism in South Eastern Missouri. *Journal of Geology.* (1937) **45**, 48-75.

[2] J. T. Singewald, Jr. and C. Milton: An Alnöite Pipe, Its Contact Phenomena and Ore Deposition Near Avon, Missouri. *Journal of Geology.* (1930) **38**, 54-66.

[3] Victor J. Blum: The Magnetic Field and the Geology of the Canyon City Area. PhD Dissertation, St. Louis University. (1944) 186.

[4] C. A. Heiland: Geophysical Exploration. 148 (1940) New York. Prentice Hall.

[5] Albert L. Kidwell: Post-Devonian Igneous Activity in Southeastern Missouri. Mo. Geol. Survey. *R. I.* 4 (1947).

Discussion*

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* TP 2978

A — Metal Mining

Diamond Drilling Quartz-Feldspar Intergrowths

by L. C. Armstrong

DISCUSSION

Burton J. Westman—Besides decreasing the diamond size, there appear to be two other approaches open to overcome excessive diamond loss and, more particularly, the rapid diamond polish that took place in this quartz-feldspar rock. First, however, a short discussion of the diamond characteristics should be made in order to discuss the situation more clearly.

The West African diamonds which we can assume were used for this drilling have two disadvantages when drilling hard, exceptionally fine-grained rocks. These are rapid polishing and, if the bit isn't "drilled in" carefully, a tendency to damage, resulting in excessive diamond loss. The normal wear resistance of any diamond is inversely proportional to the compressive strength of the diamond, therefore only the highest grade of diamonds free from insipient fractures should be employed. Using upwards of 90 diamonds per carat instead of the 20 to 30 per carat not only decreases the total projected area of the cutting points but increases the stress per point which, in turn, causes greater depth of penetration into the rock. Considering this depth of penetration, therefore, suggests an approach other than increasing the number of diamonds per carat and that is to decrease the actual number of diamond points (also the carat weight per bit) so that the penetrating force on an individual diamond is increased. This penetration increases the chipping or rock shearing action.

It should be considered that diamonds of a sufficient number that present a fairly large projected contact surface will, as in the 20 to 30 per carat size, require an exceptionally high bit force sufficient to cause a great enough "point force" to penetrate the hard, fine-grained rock and, if this weight is insufficient, the abrading action rapidly polishes the stones; therefore, by decreasing the point size, thereby increasing the point weight, penetration is affected. A similar effect may be had with the 20 to 30 per carat stones by reducing the number of diamond points so as to reduce this contact area and, at the same time, create sufficiently high point weight to affect the penetration. The latter, however, has a limit that is governed by the strength of both the diamond and the setting bond.

The polish on a West African diamond in a given hard, fine-grained rock is directly proportional to the force applied to the bit. This polish is often severe enough to retire a bit after relatively few feet of coring. To overcome this problem the Congo diamond is being employed because this type of stone does not take a polish since it wears away at a uniform rate and presents a minutely jagged surface that continues to cut for the full life of the diamond. The sizes of this diamond available, however, are seldom smaller than around ten per carat.

In general, it can be stated that for fine-grained, hard rock there are four possible approaches to reduce polish and excessive diamond loss. First is the use of numerous small diamonds, second is the use of fewer large diamonds, third is employing the Congo diamond, and fourth is the new type of crown design that has 12 to 24 small waterways, depending upon the bit size, and which has had a marked effect on reducing this polish apparently by increasing the sludging efficiency which practically eliminates the possibility of regrinding the cuttings.

L. C. Armstrong (author's reply)—My reaction is that all of Mr. Westman's contributions are based on considerable study and experience. I feel that his suggestions should prove useful to those engaged in drilling resistant, intergrowth-bearing rocks.

B. J. WESTMAN, Koebel Diamond Tool Co., Detroit, Mich.

B — Minerals Beneficiation

The Effect of Mill Speeds on Grinding Costs

by Harlowe Hardinge and R. C. Ferguson

DISCUSSION

Oscar Johnson—In my opinion, the effect of mill speeds on grinding costs must be studied along with capital investment and dollars gathered together as profits.

Comparing the entire groups of operators with those who have had the opportunity to make slow-speed mill studies, I think you will find the latter small in numbers. Most managers want the equipment worked to its maximum output. There are, however, some installations where plant and mill sizes are such that they can do the job with reduction of mill barrel speeds.

The past and the present installations of the industry are laid out to get the most capacity for the least capital outlay. This is the case even with the plants of Chile Exploration, International Nickel, Morocco, and Anaconda, now under construction or being changed. The industry recognizes that most all equipment it buys today is good and can be depended upon for efficient performance.

Under this scheme of things, I am doubtful that slow-speed ball mill operation will be generally applicable.

With reference to the U. S. Bureau of Mines laboratory tests, I think table II could have been omitted. It is inconclusive as to maximum efficiency for the low-pulp level mill on hard ore. There should be no question about this point. However, data on mill speeds can be found to substantiate various theories as well as refute them.

Gow, Guggenheim, Campbell and Coghill, in their paper on Ball Milling,[2] believe their 2 x 2 ft laboratory mill reflects results that can be expected from large mills. If so, then referring to their table II, they state, "The conclusion to be drawn from this second series is that high speed, not exceeding 72 pct of the critical, favors capacity, as before, but that with proper conditions of operation high speeds may give as good efficiency values as low speeds. In this case the efficiency values are nearly constant. A horizontal curve would indicate that the amount of grinding was directly proportional to the power expended, and these tests suggest that such a condition can be made to exist in commercial operations."

Table II

(From Paper by Gow et al)[2]

Speed, Pct Critical	32	42	52	62	72	82
Capacity:						
Surface tons per hr (65-mesh)	26.6	42.1	54.4	65.9	74.3	74.1
Surface tons per hr (200-mesh)	56.1	87.4	112.7	137.1	154.2	153.0
Efficiency:						
Surface tons per net hp hr (65-mesh)	35.7	36.3	36.3	35.4	34.3	32.3
Surface tons per net hp hr (200-mesh)	75.3	75.3	75.1	73.7	71.0	66.0
Ore in mill, Lb.	98	100	100	113	122	165

The field performance data, table III, represents much effort in its collection and preparation. But, one must realize that there are many variables that effect the efficiency of grinding mill operation, and too much must not be assumed as to the effect of some specific change. Possibly with changes in mill speed, the results might be more consistent by also a change in ball rationing, type of ball, volume of ball charge, pulp level and amount of pulp in the mill, pulp consisting, design of liner, circulating load, etc. Also, changes in ore character must be reckoned with when evaluating grinding performance.

At present the Climax Molybdenum Corp. is running at much reduced capacity. Mr. James Duggan informs me that at mill speeds of 17 rpm, they save a $0.025 per ton on liners and $0.025 per ton in power, but, if the demand for molybdenum increased, he would go back to higher speed to obtain maximum tonnage, as the values from the increased tonnage would far more than offset the one half saving at the slower speed.

The Inspiration ran a six months' test between mills running 21 rpm and 23.5 rpm. The slower mills ground 10 pct less ore with a slight saving per ton, but when the reduced plant tonnage was checked back into the actual cost figures of concentration, the high-speed mills with their greater tonnage showed considerable advantage.

To be convinced of possible practical results from the predictions in the conclusions, I think we would have to rely on the analysis of expert cost accountants to furnish the necessary proof figures.

Hardinge and Ferguson are to be commended for the work in preparing this paper. I am convinced that our Massco engineers should go into higher speeds with our equipment.

Harlowe Hardinge (authors' reply)—For one, I heartily agree with Mr. Johnson's opening statement that the effect of mill speeds on grinding costs must be studied along with capital investment and dollars gathered together as profits. It was on this basis and for this reason the paper was written.

Mr. Johnson, on the other hand, takes the position that, on the whole, low speeds are not justified from the economic standpoint, basing his principal reason on the fact that lower mill speeds cut mill capacities and hence reduce the gross income from the product produced. There is no denying this point. It is almost axiomatic. It is for this very reason that the overall advantage of lower mill speeds has been discounted and even overlooked. It was for this reason mainly that the paper was written in the first place.

It is one thing to plan an efficient operation at the outset, basing one's figures on the tonnage requirements at the time, and it is quite another to be confronted with the problem of increasing the output of an existing installation at a minimum of capital expenditure.

Economic consideration of a new installation is greatly influenced by referring to an old one. Too often, the analyst assumes that if this practice is followed in the new installation, one would not go wrong. It is just here that he may be wrong. Past practice and low capital expenditure are all too frequently given priority over the engineer's analysis of operating costs. When we are able to start fresh, we should give proper weight to other economic factors which do not exist in an old installation. It is these economic factors that make it possible to spend at the outset just a little more money and get it back in a matter of months and effect big savings for years to come.

F. C. Bond—This paper is of considerable importance in that it emphasizes a modern trend to operate ball mills at somewhat slower speeds than formerly. We have checked the data in the paper with that obtained

Table X. Effect of Low Pulp Load in Mill at Different Speeds

Critical Speed, Pct	Ore in Mill, Lb	Surface Tons Per hp-hr	Surface Tons Avg.	Loss Eff., Pct
30	75	21.4		
	50	21.4	21.30	2.74
	35	21.1		
40	75	21.6		
	50	21.4	21.43	2.15
	35	21.3		
50	75	21.8		
	50	21.5	21.60	1.37
	35	21.5		
60	75	21.8		
	50	21.6	21.63	1.23
	35	21.6		
70	75	21.7		
	50	21.7	21.70	0.91
	35	21.7		
80	75	21.9		
	50	21.9	21.90	0.00
	35			

from other sources and find ourselves in general agreement with the authors' conclusions.

Decreasing the speed of overflow wet grinding ball mills to about 50 pct to 55 pct of critical ordinarily decreases the cost per ton ground, both in metal wear and in power consumption, with certain limitations. The decrease in metal wear is usually larger than the decrease in required energy; and this enlarges the benefit to be obtained by slower speeds at the present time. In the past decade the cost per pound of grinding media and liners has approximately doubled, while the cost of power has increased only slightly, if at all, so that at present the cost per ton ground in metal wear averages somewhat more than the energy cost. Future cost trends are largely unpredictable.

In any particular installation the decision regarding the proper mill speed must be controlled by local economic factors. The increased initial cost of the larger mill required for slow speed operation must be justified by the expected life of the property, decrease in maintenance cost, etc., as well as the saving in metal wear and power consumption.

There are several points in the paper that deserve comment. The use of "surface tons" as a measure of grinding efficiency is widely questioned because it does not assign sufficient importance to the work done on the —200-mesh portion of the product. However, it may furnish an acceptable value for the "useful" work done according to the way in which that elusive quantity may be defined. It appears to be acceptable in this paper because all of the grinding tests are carried to approximately the same percent passing 200-mesh.

Another important uncertainty is the assumed amount of pulp present in both overflow and low-level discharge ball mills. The ball voids in the test mill contained approximately 75 lb of dry solids. In tabulating the data for overflow mills, the average of three runs at 200, 150, and 125 lb of solids, or 158 lb, was used. This is more than twice the amount of solids contained in the ball voids, and it might be questioned whether the usual commercial overflow ball mill contains this amount of pulp. The low-pulp level mill data are computed for loads of 100, 75, and 50 lb of solids in the mill, or an average value equal to the amount contained in the ball voids. Again it may be questioned whether a low-level mill does not contain ball voids in the upper part of the charge which are free of pulp.

Actual measurement of the amount of pulp contained in grinding mills, both of the overflow and low-level discharge types, as compared with the volume of the ball or rod voids and total volume below the discharge level, would be highly desirable. Such a measurement could be made by diverting and collecting the mill discharge from the instant the mill feed is shut off, and adding this to the pulp retained within the mill after shutdown. The results might be surprising and illuminating.

Table II is computed on average loads in the mill of 100, 75, and 50 lb. A similar table (X) is given for average loads of 75, 50, and 35 lb, which allows for some unoccupied void space in the ball charge. This table shows that the percent loss in efficiency in grinding chert at low critical speeds becomes larger as the pulp load in the mill decreases. Data are not available for the construction of a similar table on dolomite.

The data appear to justify the following conclusions regarding mills receiving relatively fine feed:

1. Considerable savings may be effected in long time operations by decreasing the speed of wet grinding overflow ball mills to perhaps 50 pct to 55 pct of the critical speed.
2. The saving will be greater in the case of overflow mills than in mills carrying a low-pulp level, and the optimum speed of low-level mills from a cost per ton ground standpoint is somewhat higher than that of overflow mills.
3. The optimum mill speed when grinding hard material is slightly higher than that for soft material.
4. Slower mill speeds may require a larger classifier for the same feed tonnage because of increased circulating load. However, this point has not been established.

R. C. Ferguson (authors' reply)—Mr. Bond questions the use of 200, 150, and 125 lb of solids in a mill containing a 796-lb ball charge as being comparable to the amount of solids in a trunnion overflow mill as the average of 158 lb of solids is more than twice the amount of solids required to fill the voids in the ball charge. Likewise, he questions the 100, 75, and 50-lb ore charge's being considered equivalent to a low-pulp level mill as the average charge of solids is equal to the amount that would be contained in the ball voids.

The only published data known to the writer on the amount of dry solids contained in a ball mill are given by E. W. Davis.[4]

This paper covers three tests made to determine the amount of pulp and the dilution in a 3 x 3 ft overflow cylindrical mill. The results found were as follows:

Test No.	Ball Charge, Lb	Dry Solids, Lb	Ratio Balls to Solids
1	1884	955	1.98 to 1
2	2006	606	3.32 to 1
3	2000	259	7.73 to 1

Referring to the ore charges in question, we have:

Ball Charge, Lb	Ore Charge, Lb	Ratio Balls to Solids
796	200	3.98 to 1
796	150	5.30 to 1
796	125	6.38 to 1
796	100	7.98 to 1
796	75	10.60 to 1
796	50	15.90 to 1

It will be noted that the ball to ore ratio of 200, 150, and 125 is in the range found in the 3 x 3 ft overflow mill and that the ratio for the 100, 75, and 50 is below the ratio found in the 3 x 3 ft mill.

Mr. Davis does not show the specific gravity of the ore, but he states that it is magnetic ore from New York State and it was being ground for concentration tests; therefore, the specific gravity of this material should not be above 3.5 and would not affect the above comparisons to any great extent.

[4] E. W. Davis: Pulp Densities within Operating Ball Mills. *Transactions.* AIME (1946) 169, 155; *Mining Technology* (May 1945) TP 1843.

OSCAR JOHNSON, The Mine and Smelter Supply Co., Denver, Colo.; F. C. BOND, Allis-Chalmers Mfg. Co., Milwaukee, Wis.

Grinding Tests on Conical Trunnion Overflow and Cylindrical Grate Ball Mills

by Jack White

DISCUSSION

W. C. McKinnon—The paper by Mr. Jack White represents a most interesting comparison. To fully evaluate the grinding tests we should have such pertinent data for the two types of ball mills as follows: mill speed, percent solids in the mill discharge, type and contour of liners (particularly shell liners), type of discharge grate used in the cylindrical mill (intermediate or low level), net open area of grate and amount of blinding during tests.

The high consumption of liner steel and balls in the cylindrical grate mill is not surprising if the mill was run at high speed and if it was equipped with a full or low-level type discharge grate.

The results would be of great interest if the cylindrical mill could be operated as an overflow type mill under the same conditions that prevailed in the grinding tests reported.

Jack White (author's reply)—As I am no longer at Mufulira I have not the data available to make positive statements but speaking from memory, the speed of both types of mills was 18.2 rpm, the percent solids of the discharge was about 70 pct. Liners were of the usual wave type in both mills. The net open area of the grates was about 45 pct and no blinding was noticed during the tests.

Before I left Mufulira, we converted the cylindrical mill to an overflow type mill, but there was not time to conduct any tests on it. No doubt these will be done in due course and it is suggested that a letter to the Mill Superintendent at Mufulira, Mr. A. A. Finn, would result in data on these tests.

Oscar Johnson—We note from Mr. White's interesting paper that all of the grinding units were equipped with the same size classifier, namely 8 x 30 ft.

When you consider the October to December operations with the tonnage and circulating loads, the classifier on the No. 10 mill had a feed of 5047 tons per 24 hr as compared to 3954 tons for units 1 to 9. Thus with about 28 pct more feed to the classifier on unit No 10, I believe this would account for the lower solids in the overflow to maintain 5.0 pct on 65-mesh against 6.4 on the other units.

If we assume 138 sq ft of classifier pool surface, the No. 10 unit had 36.6 tons per sq ft of surface as against 28.6 tons per 24 hr per units 1 to 9. I wonder what would be the comparison if the No. 10 unit had a 10-ft wide classifier for equal pool surface.

This should help on the classifier overflow product, and permit unit No. 10 to handle a larger circulating sand load with a resulting increase in unit tonnage. We would also expect some additional advantage of a 10-ft diam mill against a 9 ft based on various investigators' findings that the capacity of a mill varies as the diameter to the 2.6 power.

Mr. White has told me that the block type lining was made from cast manganese steel, the balls of forged steel. This clarifies the discussion as to the kind of material in these parts.

I called Mr. White's attention to the fact that the Marcy mill was running only 17.8 rpm (70½ pct C.S.) which is 10 pct lower than the standard recommendations. Regarding this, Mr. White writes, "your remarks about the speed and ball load are very interesting, and it is my intention to continue with the test when the opportunity presents itself, using the information that you have given me. The test, that was the subject of my paper, was conducted in 1942, and in the seven years since then the Marcy has done a good and steady job."

Jack White (author's reply)—If a larger classifier had been used on No. 10 grinding unit, I have no doubt that the results would have been different, and it is possible that they would have shown more advantage to the No. 10 mill. It is also possible that we could have run the overflow density in a larger classifier at the same solids as we did in the classifiers on units 1 to 9.

OSCAR JOHNSON, The Mine and Smelter Supply Co., Denver, Colo.; W. C. MCKINNON, Allis-Chalmers Mfg. Co., Milwaukee, Wis.

Progress Report on Grinding at Tennessee Copper Company

by J. F. Myers and F. M. Lewis

DISCUSSION

W. I. Garms—The authors state that when they added 11 tons of balls to the 45 pct volume ball load, the power needle did not budge. The question arises as to whether any increase in capacity accompanied the 11-ton addition. It is hardly conceivable that all of the balls making up the 11-ton addition were sterile. However, as the work done in the ball mill is not by kilowatthours per se or balls per se but by kilowatthourballs, it is possible that the kilowatthourballs in the mill before the 11-ton addition had a different amperage, voltage or robustage than those in the mill after the addition and that the overall outcome was that equal work was done before and after the addition.

The authors can keep the efficiency that they got with the 45 pct volume 1-in. ball charge and get additional capacity by putting more kilowatthourballs to work in the form of the addition of 11 tons of 2-in. balls to the present ball charge, and subsequent makeup additions by weight of 20 pct 2-in. balls and 80 pct 1-in. balls.

J. F. Myers (authors' reply)—Mr. Garms points his finger at the 11 tons of "sterile" balls added to the Tricone mill. A fact so interesting to us that we felt it should be reported. Clearly, ball slippage at the existing mill speed prevented any further transfer of power from the shell to the ball mass.

We agree that the kilowatthourballs would increase, were we charging rough 2-in. balls as suggested by Mr. Garms. The assumption is, of course, that we would get a corresponding increase in work done. Our reported data (CIME meeting, Vancouver, 1946) shows a gain of work accomplished of 7.6 pct as we decreased the ball size from 2 in. to 1½ in. and a further gain of work accomplished of 2.1 pct when we went from 1½ in. to 1 in. The small ball results reported by H. R. Banks at Chapman Camp are also very convincing. In the case of 2-in. balls they would take the conventional ball path and start cascading. They would thus destroy the classifying pool over the "foot" of the

rolling balls, which is the key to the overall improvement in the Tricone process.

We do not expect that students of grinding can immediately visualize the possibilities of the classifying pool caused by the small rolling balls. That could not be expected until students have directly investigated the phenomena themselves and explored its possibilities.

Our grinding process with smooth 1-in. balls has reduced by nearly one half the metallic losses in the fine micron sizes of the tailing. This is simply because less of the fine micron sizes are produced. Since the +65-mesh size is the same as formerly, a higher percentage of the intermediate sizes are developed. These sizes have the highest floatability, require the least reagents, and use less floating time.

These factors contribute so heavily to the overall economics that dropping our power grinding gain from 28 pct back to 19 pct is a small detail. However, we feel that this is only a momentary situation and that eventually the best features of the grinding and flotation processes can be brought together, which is as it should be. After all, we are not operating a quarry or gravel pit where only mesh tons are important.

Mr. Garms has very recently started operation of some big 10½-ft slow-speed ball mills with 2-in. cast balls and he reports that the ball paths are of the conventional type as planned. Hence, there exists a turbulent pool at the toe of his ball mass. He has accomplished what he set out to do kilowatthourball-wise.

This poses an interesting question. Is the selective grinding action caused by the classifying pool in the Tricone mill applicable to only ores of high specific gravity or does it have a broad application? The trick is that the percent solids in the mill feed must be relatively low to permit classification in the pool to take place. On a low specific gravity ore, such as a porphyry, can the mill dilution be lowered to a classifying point? That, nobody knows at this writing.

Mr. Garms will, of course, operate his mills as he engineered them, but we predict that he will eventually decide that if his kilowatthourball input were applied to only those ore particles in the mill that needed the power, he would be better off in the overall picture. We predict that he will order some smooth, small balls to be sure there is nothing to this selective grinding action by means of an adequate classifying pool in his ball mill. There is some basis for this prediction, as in the past, Mr. Garms has had great respect for the fact that grinding is for the purpose of preparing flotation feed and not grinding per se.

W. I. GARMS, Kennecott Copper Corp., Hayden, Ariz.

Behavior of Mineral Particles in Electrostatic Separation

by Shiou-Chuan Sun, J. D. Morgan and R. F. Wesner

DISCUSSION

O. C. Ralston and **F. Fraas**—Dr. Sun and associates have presented an interesting paper not all of which is comprehended by us. The data assembled measure the deflections of particles in an electrostatic field as a function of a number of independent variables and some dependent variables that are not sharply differentiated. These data are all based on a Johnson type machine of definite, well-described geometry, something not often done in electrostatic separation literature.

One new fact brought out by this technique is the effect of coal dust on admixed pyrite and quartz. The effects are opposite in character, as should be expected and we do not agree with the authors that these effects are negligible.

Fraas[7] also used a multiple cell "distribution analyzer" and gives in fig. 5 of his paper a straight line plot with no humps or curves. This is not necessarily at variance with Sun's results because Fraas used a larger gap between electrodes and had no evidence of particles adhering to or dropping off the charged electrode.

The section of Sun's paper on effect of surface conductivity contains a speculation that the dielectric constant "represents more or less the electrical conductance of the bulk body instead of the surface of the mineral particles." A simple picture of the meaning of the dielectric constant is that it is the specific inductive capacity of a dielectric when used as the dielectric between the plates of a condenser. It is at once evident that the above speculation confuses *capacity* with *conductance*—two definitely independent variables.

We ask the authors to state in what group or subdivision their garnet belongs; what method and units were used in calculating the data of col. A, table I and their meaning; what was the temperature of the carrier roll and, finally, has any effort been made to investigate the effects of particle shape on distribution in the electrostatic field?

S. B. Hudson—I have read this article with great interest. We have been engaged in research work on the principles of electrostatic separation in this laboratory for some time now, and our findings agree with those of the authors in many respects. The article shows evidence of careful and valuable research in the field of electrostatic separation.

A "distribution analyser," very similar to that described in an earlier article by one of the authors,[11] was incorporated in an inclined plate-type electrostatic separator designed and built in the Melbourne University laboratory in 1948 for investigation purposes.[22] The actual splitting edges were machined from ¼ in. perspex, and the paper hoppers were supported on linen thread immediately below the perspex dividers. These dividers fitted into machined slots in a framework to give accurate ½-in. spacings. The hoppers (staggered) fed directly into a rack of test tubes, which is supported on a vertical pantograph arrangement. The rack was positioned with guides on the horizontal pantograph stand, and this ensured positive alignment when replacing the rack after making weighings.

In later work, when much heavier feed rates were used, of the order of 30 to 40 lb per in. per hr a rack fitted with rectangular metal containers and similarly aligned was used.

Some work was done here on comparing the distributions of minerals when passed separately and when passed as a mixture, and it was found that there was quite an appreciable difference in the two results.[22] However, in our separator the particles do not pass down the plate in a single layer, and this difference is probably caused by collisions of one mineral particles with the other mineral particles.

In most of the investigational work here, the change of the center point of the distribution is measured to establish the effect of a variable, such as voltage. Two minerals (zircon and rutile) have been studied rather exhaustively, and it was found that their distributions are very nearly normal. Owing to the sharpness of the distribution curves, the usual method of obtaining the mean or median was inaccurate, and was not used; instead the mean (also the median), calculated on the assumption of a normal distribution, was used to locate the center point of each distribution and proved satisfactory.

The effect of polarity becomes very apparent in the plate-type separator where frictional charges play a very important part when using highly resistive minerals such as zircon. With rutile, a comparatively conductive mineral, polarity of the electrode has little effect. On the other hand, the magnitude of the voltage has a far greater effect on conductive than on resistive minerals.

Shiou-Chuan Sun (authors' reply)—Thanks are extended to Drs. Ralston and Fraas for their keen interest in this paper. Their questions concerning coal dust,

compartment box, conductance, garnet, temperature, and particle shape are answered in the same order. The effect of coal dust on the behavior of both pyrite and quartz was appreciated in this paper. The small difference between curves 3 and 4 and also between curves 5 and 6 of fig. 3 was due to that. In a synthetic mixture of 82 pct coal, 14 pct quartz and 4 pct pyrite by weight, the surfaces of both pyrite and quartz were also contaminated by coal particles. The compartment box used by Fraas, having only eight 2-cm compartments, obviously cannot be compared with the distribution analyzer of this paper. In dealing with dielectric constant, the meaning of the term "electrical conductance" is better expressed as "dielectric conductivity," which is equivalent to relative permitivity, dielectric constant, and specific inductive capacity. The garnet used was rhodolite collected originally from Macon County, N. C. The method of calculation and the meaning of the data in col. A, table I can be found in Taggart's "Handbook of Mineral Dressing." Both the grounded roll and the charged roll of the electrostatic separator were operated at room temperature. No attempt was made to determine the effect of particle shape on distribution in the electrostatic field.

It was a pleasant surprise to learn that a similar distribution analyzer was developed and successfully used in 1948 in the Melbourne University, Australia. The record shows that our distribution analyzer was first used in June 1947. The efficiency of a distribution analyzer, to a certain limit, increases with the decrease of the width of the individual cell. A micro-distribution analyzer has been designed and constructed by the senior author in the Pennsylvania State College for special research purposes. This new distribution analyzer, consisting of sixty ½ cm cells, was carefully machined from a piece of 1-in. thick lucite.

The authors wish to express their appreciation to Dr. S. B. Hudson for his interest in our paper and his additional information on electrostatic separation.

[21] Shiou-Chuan Sun: Analyser Aids Electrostatic Research. *Engineering and Mining Journal*, (1949) 150, May, 90.

[22] Council for Scientific and Industrial Research, Ore Dressing Section, Melbourne University Laboratory. R. I. No. 372, 1.

[23] Council for Scientific and Industrial Research, Ore Dressing Section, Melbourne University Laboratory. R. I. No. 372, 11.

O. C. RALSTON, Bureau of Mines, Washington, D. C.; F. FRAAS, Bureau of Mines, College Park, Md.; S. B. HUDSON, Melbourne University Ore-Dressing Laboratory, Carlton, Victoria, Australia.

Experiences with a Density Recording and Controlling Instrument for Heavy-media Separation Units

by J. J. Bean

DISCUSSION

F. M. Lewis—I believe that density recorders are a prerequisite to all well operated ore concentrators. Well designed density instruments are very accurate and give an excellent record of the steadiness of the operation. Controlling the density of a pulp with an instrument, as described by Mr. Bean, is new and the experiences in developing this controller at the American Cyanimid Co.'s Mineral Dressing Laboratory are very interesting and should be extremely helpful to others who are planning on designing or installing controlling density instruments.

Instruments for recording the density of the classifier overflow or flotation feed are rather common in ore concentrators, but there are a number of profitable uses for these instruments that are not too well known.

Density recorders with bubble tubes submerged to different depth in a classifier pool give an excellent record of the classifier operation. The loading and unloading of the machine, that is so easily masked in the density of the overflow, can be easily recorded if the instrument is installed at some depth in the classifier pool. On some occasions, two instruments with bubble tubes at different depths give a more candid picture of the irregularities in the classifier operation.

When cleaner machines are operated in closed circuit with roughers, a record of the density of the cleaner tails will indicate the circulating load in the circuit and is very helpful in coordinating different operators, because quite often the circulating load will be increasing for some time before the analyses of the products will change.

Density recorders are used to record the variations in the dilution of the underflow from thickening tanks and in at least one plant the instrument controls the underflow.

F. M. LEWIS, Assistant Superintendent of Mills, Tennessee Copper Co., Copperhill, Tenn.

Some Recent Investigations with the Dutch State Mines Cyclone Separator on Fine Coal Slurries

by A. A. Falconer

DISCUSSION

D. A. Dahlstrom—Mr. Falconer has done an admirable job of proving the adaptability of the cyclone to the beneficiation of a very difficult size range in the preparation of coal. The addition of the cyclone to other coarser size methods makes it economically feasible for the coal operator to rigorously control his marketed products at a high and uniform quality level down to the 48 and 100-mesh size. Furthermore, it should be pointed out that by utilizing the cyclone as a deslimer or classifier the −200-mesh fraction, usually containing large percentages of undesirable material, can also be quickly and cheaply removed from the clean coal. Advantages of such a step are several: (1) Reduction of ash content, especially where clays and slimes are serious. (2) lower moisture content of the final coal. (3) easier mechanical and thermal dewatering of the fine coal due to higher cake and bed permeabilities.

The author has indicated that adjustable control means should be installed on the cyclone operating on a heavy-media slurry. This also holds true for the cyclone acting as classifier, deslimer or preliminary dewatering agent and is worthy of special emphasis. Such controls, while not necessarily automatic, must be simple, rapidly adjustable, and require a minimum of attention. Three methods available to the operator are (1) throttling valve on the feed line, (2) back pressure valve on the overflow, and (3) adjustable diameter underflow nozzles. The first two, while fulfilling the specifications, suffer the disadvantage of restricting capacity if severe throttling is necessary. The third method has little effect on capacity and easily maintains an underflow of correct moisture content. Several designs are possible, but two find the widest application. The first was developed by Robert Piros of The Truax-Traer Coal Co. and uses a simple casting containing two holders for underflow nozzles which can be rotated about an axis parallel to the cyclone centerline. These holders retain the underflow nozzle flush with the conical section walls. When a change in nozzle diameter is required, the new size is placed in the spare holder and rotated into place. A second design utilizes a soft rubber cylindrical tube which is compressed or released from three sides, causing the inner diameter of the tube to decrease or increase. This is achieved by using a machine-threaded,

perforated cap on the cyclone underflow pipe which acts on a compression ring to increase or decrease the force on the rubber tube. The cap is fitted with horizontal side arms to facilitate easy adjustment.

D. A. DAHLSTROM, Northwestern University, Evanston, Ill.

Concerning the Adsorption of Dodecylamine on Quartz

by A. M. Gaudin and F. W. Bloecher, Jr.

DISCUSSION

G. L. Simard and **D. J. Salley**—The authors and ourselves[12] independently came to similar conclusions both as to the value of tracer methods for the study of flotation and the general nature of collector-mineral interaction. The dodecylamine-quartz system appears to be simpler than the dithiophosphate-galena one, and the results therefore appear clearer. It is evident from a comparison of the two papers, however, that care must be used in extending the results to other more complicated systems.

In the case of dodecylamine-quartz, the possible influence of micelle formation is of interest. There is no à priori reason why complete monolayer formation should necessarily occur at the concentration for micelle formation, since sorption depends on the nature of the substrate as well as on the properties of the solute in solution which alone determine micelle formation. Indeed, the fact that the onset of increased adsorption, point *B* in fig. 2, occurs at a concentration one tenth the critical value for micelles might imply that micelle formation does not have an influence. Interestingly enough, the data may be handled by the methods used for multilayer adsorption of gases on surfaces. Thus, if the amount adsorbed is plotted versus concentration, a sigmoid type curve is obtained, similar in form to a so-called Type II isotherm[13] generally attributed to multimolecular adsorption. A monolayer value of about 0.12 mg per g may be estimated from the knee of this isotherm. On the other hand, if a plot is made according to the multilayer gas equation for adsorption to an unrestricted number of layers, eq 38,[13] assuming the critical micelle concentration as the saturation concentration, the data above 120 mg per liter fall on a straight line from which a monolayer adsorption of 0.3 mg per g results. These considerations suggest that the monolayer may occupy an area somewhat less than the geometric value of 0.4 mg per g, and that micelle formation may indeed be a factor in the adsorption.

[12] G. L. Simard, J. Chupak and D. J. Salley: Radiotracer Studies on the Interaction of Dithiophosphate with Galena. *Transactions* AIME (1950) **187**, 359; *Mining Engineering* (March 1950) TP 2815.

[13] S. Brunauer: The Adsorption of Gases and Vapors. **I**, Princeton University Press (1943) 149-162.

G. L. SIMARD and D. J. SALLEY, American Cyanamid Co., Stamford, Conn.

Measurement of Equilibrium Forces Between an Air Bubble and an Attached Solid in Water

by T. M. Morris

DISCUSSION

H. H. Kellogg—There is one point that the author has failed to emphasize sufficiently in his paper.

What is commonly called the equilibrium contact-angle (the author's "maximum contact-angle") can have only one value on a smooth, flat, homogeneous surface under a given set of conditions. The equilibrium contact-angle is defined, for such a system, as the angle between the solid-liquid and liquid-gas interfaces, measured through the liquid. The value of the equilibrium contact-angle is uniquely determined by the value of the interfacial energies of the three intersecting interfaces and is independent of other forces in the system.

When the liquid-gas interface intersects the solid at an edge—as was the case for all the experiments reported in this paper—the orientation of the solid-liquid interface is indeterminate or varies through 90° for a right-angle edge. Mr. Morris has called the angle between the horizontal and the liquid-gas interface for this edge condition a "static contact-angle." I feel that this term is unnecessarily misleading. In the first place, "static contact-angle" sounds too much like "equilibrium contact-angle." In the second place, the magnitude of the "static contact-angle," which I would prefer to call the "supporting angle," is determined by the forces in the system other than those derived from the interfacial energies, hence "contact angle" is misleading. If Mr. Morris had said that the "supporting-angle" is variable and depends on the weight of the particle and size of the bubble and that it has a maximum possible value equal to the equilibrium contact-angle, his discussion would have been more accurate.

T. M. Morris (author's reply)—Mr. Kellogg puts forth a reasonable criticism of some of the terminology used in the paper. I agree that the term "static contact angle" may be misleading. Substitution of the term "supporting angle" or "vector angle" may be more suitable.

F. X. Tartaron—In this paper the author presents a very interesting mathematical development of the forces present when a mineral particle adheres to an air bubble. Excellent concordance is obtained between mathematical formulation and experimental results.

It is the writer's understanding that when the mineral surface presented to an air bubble is greater than the area of contact, the maximum contact angle is obtained. However, this contact angle represents distortion of the bubble, the normal shape of which is spherical or in cross-section, circular. This distortion produces a force that acts in opposition to the force of adhesion between the bubble and particle. Hence, at maximum contact angle, the force of adhesion between bubble and particle is at a minimum for static conditions. However, when the size of bubble is increased, the size of particle remaining the same (and all other conditions remaining the same), the contact angle decreases, the distortion of the bubble decreases and the force in opposition to adherence of bubble and particle also decreases. Thus, there is stronger attachment between bubble and particle.

When this situation is applied to actual flotation conditions, it is doubtful that it has any significance. The reason is that the bubbles are normally so much larger than the particles, that in substantially all cases, it is probable that negligible distortion of the bubble takes place. In table IV, the author makes computations for bubbles from 0.50 to 2 mm diam. This is from 0.02 in. to 0.08 in. Certainly the bubbles generated in a flotation machine are far larger than this. The surfaces of the author's bubbles range from 0.79 to 12.6 sq mm. The area of contact of the glass rod (0.15 mm diam) is 0.017 sq mm. Thus, ratio of bubble surface to mineral area of contact ranges from 47 to 741 in round numbers. If we take a ¼-in. diam bubble and a 65-mesh (0.208 mm) particle of cubical shape, the ratio of bubble surface to mineral area of contact is 2931.

Mineral particles do not readily become attached to air bubbles. Taggart has shown that in pneumatic flotation machines collector-coated particles are only temporarily attached to bubbles. They keep falling off and down in the froth but at a delayed rate as compared with gangue. Spedden and Hannan's motion pictures confirm the difficulty of attaching particles to air bubbles. In the agitation froth process, according to Taggart, air is precipitated from the water selectively on to the collector-coated particles and these

bubble-mineral aggregates coalesce to form the mineral-bearing froth. However, in this case, it is difficult to see how the vast quantity of air utilized can dissolve in the water and be precipitated in so short a time. It appears to the writer that in the agitation froth process, the air precipitated onto collector-coated mineral particles facilitates attachment of "coursing" or nonprecipitated air bubbles to the particle. Coalescence takes place by attachment of particles with a small amount of air precipitated onto their surfaces with large bubbles that are pumped into the pulp by the agitator in the flotation machine. Thus, in the agitation froth process, a relatively small quantity of air would be required for precipitation. The nonprecipitated air pumped into the machine in large volume serves to gather up the particles and provide them with the necessary buoyancy. It would make an interesting investigation to verify that large air bubbles attach readily to particles with minute air bubbles on their surfaces.

T. M. Morris—Mr. Tartaron states that when a bubble is attached to a mineral surface of large extent, the force of adhesion is at a minimum when the contact angle is a maximum. Just the opposite is true—the force of adhesion is at a maximum when the maximum contact angle obtains, for a given size of bubble. When the size of bubble increases, other conditions remaining constant, the degree of flexibility increases and hence such a bubble-particle system can better withstand disruptive forces than if the bubble were smaller because the internal pressure of the bubble is less for a large bubble than for a small bubble.

I can't agree that all bubbles in a flotation cell are so large that the influence of the size of bubble is insignificant. Surely ¼-in. bubbles are not the smallest size present. It is reasonable to expect that there is a wide range of sizes present in a flotation cell. Observation confirms this. The lower size would probably be smaller than 0.50 mm.

I agree with Mr. Tartaron that it would be interesting to study the influence of precipitated bubbles upon the attachment of particles to larger bubbles.

H. H. KELLOGG, Columbia University, New York; F. X. TARTARON, Jones and Laughlin Steel Corp., Negaunee, Mich.

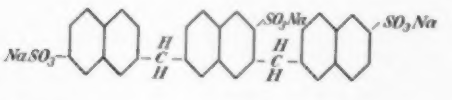
Effects of Activators and Alizarine Dyes on Soap Flotation of Cassiterite and Fluorite

by R. Schuhmann and B. Prakash

DISCUSSION

Maurice Rey—It may be interesting to note that depressing effects can also be obtained from cyclic compounds other than dyes.

One such compound which is a dispersing agent for carbon, pigments and other compounds is known in France by the trade name "dispergine" and in the



United States by the name of Lomar P W (Jacques Wolf & Co.)

Dr. Rey admits that the compound is not very selective and is similar in its action to starch and dextrin.

MAURICE REY, School of Mines and Metallurgy, Paris, France.

Continuous Countercurrent Decantation Calculations

by T. B. Counselman

DISCUSSION

C. G. McLachlan—In the foregoing paper the author has presented a very neat method for calculating the solution recovery for a countercurrent flowsheet. He has, however, based his calculations, as he states, on the assumption that "the concentration of dissolved value must be exactly the same in the overflow and underflow of any thickener." This assumption, which is usually made regardless of the type of calculation used, is not strictly correct because diffusion of the dissolved gold in the solution associated with the solids as it progresses through the countercurrent system is far from being instantaneous, with the result that on a ton for ton basis, the assay of the solution in the thickener underflow is higher than in the corresponding thickener overflow. An agitator introduced into the flowsheet between thickeners will reduce, but not entirely eliminate, this difference. In practice we have found that the best way to take care of this condition—in addition to the use of one or more intermediate agitators—is to circulate 20 pct or 25 pct more solution than called for by the cheoretical calculation. This presents no difficulty provided that adequate thickener capacity is provided in the original design of the countercurrent circuit.

C. G. MCLACHLAN, Noranda Mines, Ltd., Noranda, Quebec.

Preliminary Report of Massco Circuitron

by A. E. Craig, W. J. Tait, and E. P. McCurdy

DISCUSSION

C. M. Marquardt—The problem of the automatic control of a grinding-classification circuit is not nearly as simple as has been indicated and it cannot be universally solved through the application of this device.

The sound emanating from a ball mill such as the metallic clink of the balls against the liners can at times be misleading. The fact that the sound level due to the balls hitting the liners may be at a minimum does not necessarily mean that the mill is properly loaded. I have in mind an ore we treat that has a soft talcy gangue. The soft fluffy nature of this material makes the sound from the ball mill a most misleading parameter that is worthless as a means for controlling the amount of ore fed to the mill.

In every milling plant there are transportation lags due to the time it takes the ore to travel from the mill bins to the ball mill. In many plants there may be several different transportation lags due to varying distances of bins from ball mill. These transportation lags can most seriously affect the operation of a controller. It has been found that under many circumstances the transportation lags can be so bad as to cause the controller to hunt seriously. Hunting can and in many cases does become so serious as to render automatic control impossible because of the surges it causes. While it is possible to compensate for this type of lag, this apparatus does not lend itself readily to accomplish the required compensation. Further, if there are groups of bins at various distances from the ball mill, compensation for one may not be sufficient or may be too great for another.

Each milling plant presents a separate problem. The apparatus as presently described does not appear sufficiently flexible to accomplish this.

It should be noted that the load on the classifier rakes is determined by measuring the current input to the rake drive motor. Variations in line voltage will

also cause a variation in the current input to the rake drive motor. It has been my experience that in many places the changes in rake load that are used for control purposes are of about the same magnitude as the line voltage fluctuations. Each may cause the same change in current to the rake drive motor. One, however, has nothing whatsoever to do with the load on the rakes. The actual power input of the rakes is a much better measurable parameter for control purposes. Plants subject to much voltage variation must consider this factor.

With some persons working on this problem there is some doubt as to the value of the load on the rakes as a control parameter. Are we interested in the load on the rakes or more particularly are we interested in the kind of job of classification being done? There is no doubt that the man on the rakes has some effect on the kind of classification job being done. I seriously doubt that it is a cure-all. Further, where the mineral content of the ore varies over wide limits, the load on the rakes can be misleading, even useless, as a control parameter.

As the ore fed to the ball mill is varied, the water added to the scoop box should also be varied. For proper operation of this device another control on the water line operating in parallel with the belt control should be used to vary the water with the ore.

A. E. Craig (authors' reply)—The comments offered by Mr. Marquardt are very interesting and bring up several points which have been included in the development of the Massco Circuitron. Certainly it was not our intention to suggest that the control of the grinding circuit was in any way simple as it has involved several years of effort on our part. We agree also it probably cannot be universally used, but we maintain that there are a great number of instances in which the device can pay for itself in added tonnage and improved metallurgical results.

In installing the Circuitron, the grinding circuit is set to optimum results as near as can be determined by manual operations. The instrument is then tuned in, in a manner similar to that of tuning in a living room radio, to select the combination of classifier load reaction and mill sound which gives the best results in tonnage and grind. It took a long time and a serious effort to control this selection so that we can now definitely pick up the desired sound, eliminating such extraneous sounds as that produced by the gear and other machines away from the ball mill. In the same manner we select just the range of sound which is needed for best control. In this way we make compensation for the difference in the character of the ore, whether soft or hard, whether fluffy or otherwise.

In a direct closed circuit with the ball mill and classifier, the transport lag does not appear to be great enough for serious consideration. This condition might be troublesome in an installation dealing with two-stage grinding running the rod mill open circuit followed by the ball mill and classifier circuit. We have not yet reached this stage, but it is our opinion that the device can be adjusted to give satisfactory performance although some additional engineering will be necessary. We have had no noticeable difficulty due to holding or, on the other hand, from heavy surges. The Circuitron maintains control, changing as required by the feed changes.

The question of flexibility is largely answered by our combination method of tuning the sound circuit in connection with the adjustment of the classifier power current to the greatest advantage.

The question is raised that variation in line voltage on the classifier motor circuit might cause enough variation in the current delivered to the Circuitron so as to upset our control. As a matter of fact, in the controller circuit, the measure of the classifier power is made by a modified watthour meter, using a current transformer for the reduction of the classifier current to any desirable value. The actual current delivered to our instrument is small and we have no difficulty at this particular point. We have had a great deal of difficulty with voltage fluctuations within the control itself so that it has become necessary to use a voltage regulator to insure a constant potential of 115 v in the controller circuit.

We are indeed interested primarily in the work of the classifier, measured by the characteristics of its overflow product, rather than in the amount of sand load, although we do emphasize the fact that the sand load is one point of control in the Circuitron. The work of the classifier is maintained at a high efficiency level by adding the Massco density controller to maintain a constant density in the overflow. This results in good classifier operation on the various tonnages as the density controller is set to hold the density at a predetermined point. The last point mentioned by Mr. Marquardt is controlling the amount of water added to the scoop box. We originally figured that this amount was so small relatively that it needed no control. Later results indicate that this may be an important point and, if so, we plan to make provision for this additional regulation.

C. M. MARQUARDT, Combined Metals Reduction Co., Salt Lake City, Utah.

An Improved Method of Gravity Concentration in the Fine-Size Range

by Arvid Thunaes and H. Rush Spedden

DISCUSSION

R. R. Knobler and **F. E. Albertson**—Following the testwork done by Thunaes and Spedden, a Sullivan deck plant was built for the Colquiri mill. This plant started to operate in April 1945 and continues in successful operation.

Since May 1950 the Colquiri mill has operated with a completely revamped flowsheet at a rate of 1000 tons per day. The old Sullivan deck section was retained with this new flowsheet.

The "slime plant" in Colquiri consists of one 12-cell Denver No. 18-Sp. Sub-A flotation machine, one 16-ft and one 12-ft hydroseparator, six five-deck Sullivan-frames and six shaking tables for cleaning up the Sullivan deck concentrates, as well as 12 Deister plus three Plat-O tables on which the fine sands, which are not treated on the S-decks, are concentrated.

Only —325-mesh fines are treated on the Sullivan decks. The combined hydroseparator overflows are fed to the S-decks.

From 1946 to 1949, i.e. with the "old" flowsheet and an average mill feed of 800 tons per day an average of 45 tons per day was treated on four Sullivan rougher frames. The two additional frames were used as cleaners. At times, up to 90 and 120 tons of slimes were treated successfully on this Sullivan deck installation.

The average operating data for the S-deck section in Colquiri for the year 1949 were as follows:

	Assay Sn, Pct
Feed to S-decks	2.13
Tails from S-decks	1.03
Table cleaner concentrates	24.4

The tests reported in this paper indicated a tin recovery of 42 pct. The actual plant results for 1949 show a 54 pct tin recovery from the Sullivan decks.

The total average quantity of tin produced from the S-deck section in 1949 was 15 tons of tin per month. This corresponds to 3 pct of the total mill recovery.

The difference between the 6 pct overall recovery-increase expected after the tests in 1943 and the actual plant results given for 1949 is explained as follows. According to the tests, a total of 170 tons was expected to be routed to the Sullivan decks, but because of certain changes in the flowsheet and mill amplifications,

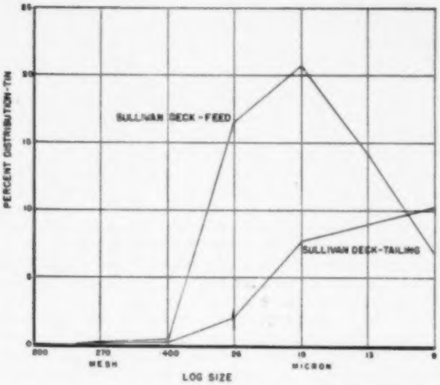


Fig. 5—Percent tin distribution in Sullivan deck feed and tailing, Colquiri ore.

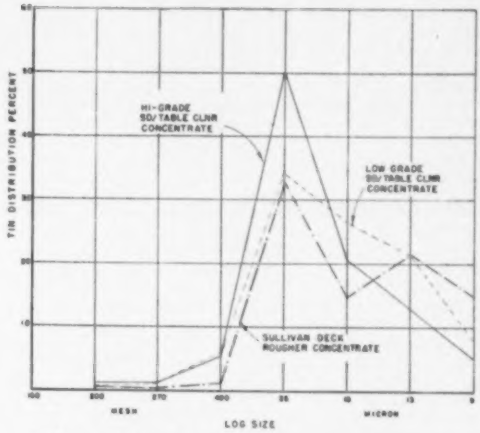


Fig. 6—Tin distribution in Sullivan deck rougher and table cleaner concentrates.

only about one fourth of this tonnage was actually treated on the decks.

With the new flowsheet a higher proportion of fines is produced and more feed is being routed now to the Sullivan section.

The tin distribution of the Sullivan-deck feed and the Sullivan-deck tails is given in fig. 5. These analyses were made on products from the pilot tests for the design of the new flowsheet. They show the high concentrating efficiency of the Sullivan decks down to about 9 micron particles of heavy minerals.

Fig. 6 gives the tin distribution of the corresponding concentrates. These concentrates showed the following assays:

	Assay Sn, Pct
S-deck concentrate	3.2
Cleaner table high-grade conc.	45.6
Cleaner table low-grade conc.	13.2

The Sullivan deck operation is extremely simple and requires very little maintenance and operating expense.

The operation of the whole slime plant in Colquiri, including flotation and fine sand tabling, cost $0.07 per ton milled.

In the year 1949 only $317.00 was spent for spare parts and material for the Sullivan decks in Colquiri with a total of 268,600 tons milled.

Although round tables of the Anaconda type recover the same sizes of fine cassiterite as the S-decks, the cheapness and ease of operation of the decks make this machine far superior.

Recently laboratory and mill tests were made with the Colquiri ore to determine the advantage of dispersing the feed to the slime concentration. The results indicate that deflocculation with sodium silicate is helpful for obtaining better hydraulic classification and for obtaining better results in the shaking table cleaning of the S-deck concentrate.

It is planned to use dispersion with sodium silicate in the Colquiri mill for preparing the S-deck concentrate for the table cleaning.

The recovery in this table operation could be increased from 42.9 pct to 63.3 pct by treating deflocculated pulp; at the same time a higher grade concentrate is obtained (21.0 pct Sn against 16.5 pct Sn).

By retabling, these concentrates are graded up to over 40 pct Sn.

The use of pine oil was also found of some advantage on the feed to the Sullivan decks. It works in the way indicated by Thunaes and Spedden and lowers somewhat the tin content of the S-deck tails.

At present tests are being made in our experimental station and in the mill trying to use the Dutch State Mines (D.S.M.) cyclone as classifier and deslimer for the slime plant in Colquiri.

Such an application of the centrifugal cone was suggested by Professor Spedden to one of us early this year.

The results obtained so far indicate that it will be possible to make the sand-slime separation at about 270/325-mesh with two D.S.M. cones working in series. This will make it possible to route only fine material to the hydroseparators and to make their operation considerably simpler. It is planned to discard then the finest slimes (after dispersion) with the hydroseparator overflow which will reduce the tonnage to the S-decks. Treating more granular material will make the deck concentration also more efficient. Instead of the hydroseparator overflows, as at present, their underflows will be treated on the S-decks.

Tungsten Slime Concentration

In 1945 we made tests on the slime tailings from the Bolsa Negra tungsten mill in Bolivia, which is also operated by M. Hochschild, S.A.M.I.

This mine contains wolframite plus scheelite in a proportion of about 9 : 1.

On a slime table tailing of 1.94 pct WO3 a Sullivan-deck preconcentrate of 7.65 pct WO3 was made which contained 49.6 pct of the tungsten formerly lost in this tailing.

Of the tungsten fed to the S-deck, 67.6 pct was +9 micron so that a recovery of about 70 pct on the —270 +1500-mesh wolframite is indicated. The S-deck tail assayed 1.15 pct WO3 and contained mostly scheelite slimes —9 micron.

Because of the high tourmaline content in this ore it was difficult to grade the 7.6 pct WO3 preconcentrate up to over 29 pct WO3.

The Bolsa Negra mill was closed shortly after these initial S-deck tests were made. Under present conditions operations at this mine might be resumed and it is planned to include a Sullivan-deck section in the new mill.

R. R. KNOBLER and F. E. ALBERTSON, Mauricio Hochschild S. A. M. I., Oruro, Bolivia.

F — Coal

Coal Preparation for Synthetic Liquid Fuels

by W. L. Crentz, J. D. Doherty, and E. E. Donath

DISCUSSION

Aureal T. Cross—This paper, which treats certain phases of the research and development of newer uses for coal, serves to emphasize the types of problems which will need to be solved and can be solved better by coordinated work of chemists and engineers with coal geologists and paleobotanists. It serves as evidence for the need and value of coal petrographic studies, paleobotanical investigations into the origin, distribution and alteration of coal, and for coal geochemical research.

It is necessary to accept the fact that coal is no longer to be viewed as a finished product as mined but as a raw material to be prepared in various ways for wise and economically sound use. We must anticipate and promote new methods of utilization, some of which may seem fantastic or wishful today. It is true that many of the more important methods to which coal is put to use today will be with us for some time, i.e., coke making, steam, heating, etc. But newer uses will probably eventually consume as much or more coal than our present economy and technologic development allows. We have really never gone far from the "cream of the crop" coal seams to supply our needs. It should not surprise us when coal technologists develop methods of utilizing even low rank, high ash coals, or such cast-off materials as the inferior "middling" products of the hydrogenation process, and even bituminous beds of very low Btu rating. We should look forward to the day when methods of utilization are so developed that each coal type can and will be put to the wisest use in order to conserve our resources. We should begin now to consider all bituminous beds of any sort as so many units of energy, or by some other such common denominator, rather than as tons of coal or gallons of fuel. We might then say "so many energy units are available in a certain form, can best be utilized for a particular job, and, by certain preparation methods at costs previously determined in laboratories and pilot plants, can compete favorably physically and economically in a certain job with other types of energy-bearing resources.

The solution of these problems hinges in part on the availability of an advance guard of coal geologists trained to observe and interpret the character of the coal components and their effects upon, reaction to, and role in chemical and physical reactions.

Best uses of the various types of coals are generally predictable by the application of various techniques of the paleobotanist and coal petrographer with reasonable accuracy at costs probably considerably lower than awkward attempts to work out some of the more basic information by chemists. These same researchers also must know their own limitations and turn over to proper specialists problems beyond their fields.

Typical problems which can undoubtedly be solved by coordinated efforts of coal geologist, petrographers, paleobotanists, coal geochemists and others are reduction of even greater amounts of sulphur both mineral and organic, a method for utilization of "fines" or even impalpable coal dust, which are now essentially waste products, assistance in roof support problems, more accurate determination of reserves, and continuity of beds and general correlation.

We should broaden our horizons but we must have the research men to pave the way. They are generally unavailable today. The fuel technologists need the basic and fundamental information that is designated by some at the present time as irrelevant, uneconomical and of academic value only. But if we insist on binding ourselves to present technologies, coal geologists of most sorts will have limited usefulness. Advance designs of old machines, combustion chambers and power converters should not furnish limiting boundaries of our vision in fuel technology.

AUREAL T. CROSS, West Virginia Geological Survey, Morgantown, West Va.

Cyclone Thickener Applications in the Coal Industry

by M. G. Driessen and H. E. Criner

DISCUSSION

Maurice Rey—The influence of cyclone diameter upon the fineness of separation is an important point which, however, cannot be discussed adequately if the injection pressure or the rate of flow are not specified.

Perhaps some light can be thrown on the matter by using the formulas derived by Dahlstrom.[5]

Let us consider several geometrically similar cyclones where the apertures *e* and *b* are equal and proportional to the diameter *D* and which are operated with the same pressure drop.

Eq 8 of Dahlstrom's paper can be written:

$$\frac{Q}{\sqrt{F}} = K\ D^{1.8}$$

or, if F is constant:

$$Q = K\ D^{1.8}$$

K is a constant of proportionality.

Introducing this value of Q in eq 13 gives elimination efficiency $= K\ D^{0.41}$
Elimination efficiency, in the terminology of Dahlstrom, means the diameter of particles, in microns, which divide equally between the underflow and the overflow. This denomination is regrettable and it would be preferable to call it diameter of separation or characteristic size or any other name connoting a size rather than an efficiency.

Whatever the name used, the above equation shows a slow increase in the fineness of separation when the diameter of the cyclone decreases.

For example, if a 14-in. cyclone gives a 400-mesh separation (37 microns), geometrically similar cyclones, having the same pressure drop will give separations at the sizes shown below.

Cyclone Diameter, in.	Diameter of Separation, Microns
14	37
8	29
6	26
4	22
3	20

Now, it seems to me that the curves of fig. 3 of Driessen show a stronger influence of the cyclone di-

ameter than the one computed above. However, the pressure drops of the various cyclones studied are not given so that it is not possible to reach definite conclusions.

H. E. Criner (authors' reply)—Mr. Rey's observation as to the rate of increase of particle size with cyclone dimension is not quite correct since the experimental data from which the exponent 0.41 was secured was not from geometrically similar cyclones but from cyclones generally having the same body diameter with varying feed and overflow opening diameters. However, the exponent is close to being correct. We believe it to be 0.5.

The theoretical relationship between particle diameter and cyclone dimensions is:

$$Dp^2 = K \frac{Aj^2}{l} \sqrt{\frac{Cf}{P}}$$

where Aj is the area of the feed nozzle, l is the cyclone core length from overflow tube to cone apex. Cf is the flow coefficient and P is the inlet pressure.

The curve referred to by Mr. Rey was meant, unfortunately, to show the trend or the separation size with respect to cyclone diameter rather than an exact relationship between them. The 1⅛ and 2 13/16-in. cyclone were similar, the 8 in. had a 14° cone angle and the 14 in. a 20° angle so that the cyclones were not similar to each other or the two smaller ones. Furthermore, the results on the 14-in. cone were secured with a higher feed concentration than that used to test the smaller cones. All tests were run at 40 psi inlet pressure.

We are sorry that we cannot furnish Mr. Rey with data from a closely controlled test, however, from the data available to us; after correction for inlet pressure, feed concentration and flow ratio; we have concluded that the particle size varies as the ½ power of the cyclone diameter.

MAURICE REY, School of Mines and Metallurgy, Paris, France.

Kerosine Flotation of Bituminous Fine Coal

by L. E. Schiffman

DISCUSSION

W. J. Parton—Those operators faced with the problem of treating fine coal whether in bituminous or anthracite will find this paper most timely.

I would like to take this opportunity of discussing Mr. Schiffman's paper and at the same time express certain views relating to our Tamaqua plant.

I would like to ask the author what type of impeller and diffuser is used in the Denver cells?

Screen analysis of products from individual cells indicate that coarser material resists flotation and only floats after greater retention time in the last few cells. Also, the need for a scavenger screen to reclaim nonfloated coal particles further stresses this point. I have always felt that more efficient means of cleaning coal between 10-mesh and 28-mesh existed than flotation. Reagent and power costs are high for the flotation process.

When floating +28-mesh particles, cell capacity is lowered and some of the particles are lost with the refuse.

The Tamaqua plant of the Lehigh Navigation Coal Co. floats —28-mesh coal and capacity of recoverable coal is 40 tph for 1800 cu ft of Denver cells; or 0.05 tons per cu ft of cell.

At Kimberly 7.75 tph for 600 cu ft of cell gives 0.013 ton per cu ft of cell.

At Bessie 14 tph for 800 cu ft of cell gives 0.017 tons per cu ft of cell.

It would be appreciated if the author would comment on what he feels is the upper size limit of particle to attain most efficient utilization of the flotation process.

The dewatering screw is a very interesting development since it offers a simple way to prepare coal sludge for more complete dewatering by drainage or mechanical dewatering on screen or filters. In other words it could be used to accomplish the same thing as a thickener tank. I would appreciate having the author's comment on how he thinks such a screw dewaterer would work on a froth.*

The process as used in floating coal at the Bessie and Kimberly plants may be referred to as more of a bulk oil float in contrast to a froth flotation process.

Experiments on increasing capacity of cells is most interesting since we are going through such an experimental period at the present time. Recently a double overflow was installed on our No. 30 Denver cells. So far results are not conclusive.

In reviewing this paper the following comments are made pertaining to investigation of methods for increasing capacity:

Supercharging: Supercharged air in matte flotation or for that matter the use of the normal amount of air drawn in by the impeller would in all probability cause such an aeration in the cell as to destroy the buoyant effect given to the coal particles by the excessive amount of kerosine used.

In other words, air creates an agitation zone throughout the cell, creating a boiling and thereby giving a lower recovery in the cell. It would be interesting to know whether the 7 pct increase in recovery was with no air being admitted to the stand pipe.

Changing Impeller Speed: The speed of a receded disc impeller for a No. 30 cell as recommended by the Denver Equipment Co. is, I believe, approximately 250 rpm. At this speed and using supercharged air in excess of 8-oz pressure, we have observed a boiling action in the cells. In our flotation we endeavor to obtain some degree of froth flotation using pine oil as a frother. The boiling action as caused by increasing the amount of air added to the cells is detrimental to recovery in froth flotation.

It is our belief that to obtain increased recovery from a cell in froth flotation, additional air must be introduced but at the same time this air must be dispersed throughout the pulp in the form of small bubbles and this can only be done by increasing the speed of the impeller.

Therefore, if Mr. Schiffman decreased the speed of the No. 30 impellers and at the same time continued to use supercharged air, the boiling action may have been increased because larger bubbles developed. The lower recovery as reported could be due to this factor.

Decreasing the impeller speed will definitely decrease the power consumed but may have other disadvantages. First, we believe it will permit "sanding" in the cell and this in our opinion will increase the wear on the impeller and diffuser, especially so, if there is pyrite and/or sand present in the feed.

"Sanding" in the cell when air is used, as in froth flotation, will effect the dispersion of this air and cause boiling.

* Concentrate rather than a matte concentrate.

When one considers the number of particles of coal to be floated and the maximum load each bubble of air can lift, it would seem to be imperative that greater increase in the number of bubbles is required. In matte flotation the more coal to be floated, the more kerosine required, which accounts for the high reagent consumption as compared to our Tamaqua plant which uses about 0.9 lb fuel oil per ton feed.

As stated previously, the use of double overflows on the No. 30 cells at our Tamaqua plant are being investigated. To date the increase or expected increase in capacity has been disappointingly low. Why? It is our belief that employing an impeller speed of 250 rpm as recommended, the amount of air being introduced is being fully utilized, the same as it was with a single overflow, and therefore no additional recovery can be obtained with the double overflow until we increase the impeller speed to allow more air to be introduced into the cell in the dispersed state.

There is also the possibility that when the cell volume was increased, the agitation was decreased, which would permit "sanding" and help to destroy the good flotation conditions which previously existed.

It is interesting to note that in the flotation of phosphate in Florida, our plant reports doubling the capacity of their cells using more air and increased impeller speed. They also do not have as many particles to float per unit volume of pulp.

Increased Area and Coal Removal. When the effective area of a cell is increased, the use of rakes, especially the use of split rakes in place of a double spitz, for matte flotation of coal or the use of double overflow paddles in froth flotation, to effect an increase in the rate of coal removal (increased capacity) would seem to be a logical conclusion providing *more* coal is raised to the surface by changing conditions in the cell proper.

Conditioning. Conditioning prior to flotation would in all probability decrease the retention time at each of these plants and for matte flotation there is a possibility some saving of kerosine would result if conditioning were done at a higher pulp density. The flotation cells are conditioners. Lowering the impeller speed to 207 rpm at the Kimberly plant as reported by the author resulted in the first cell only floating 80 pct of its normal amount and the second cell only 90 pct. This could be attributed to a decreased conditioning action. Insufficient conditioning in the cells could result in a loss of kerosine with the coal product with a resulting increased loss of coal in the refuse.

In summary it is our belief that many factors enter into the problem of increasing the capacity of a No. 30 Denver flotation cell.

L. E. Schiffman (author's reply)—The impellers used in the Denver cells are the receded disc type.

Our experience would indicate that 20-mesh is probably the top particle size for the most efficient utilization of the flotation process, but that the top size can be raised to 14-mesh with but little impairment in efficiency. Above 14-mesh there may be considerable impairment, but the need for cleaning coarser than 14-mesh, the simplicity of the kerosine process, and the avoidance of an additional cleaning method led to the selection of 10-mesh as the proper size for Kimberly.

It is true that our floated coal capacity expressed in tons per hour per cubic foot of cell volume seems low in comparison with other flotation processes. According to Mr. Parton's figures the float at Kimberly thus expressed is but 26 pct of that obtained at Tamaqua, and at Bessie but 35 pct. We have always attributed our lower results to the lighter gravity of the material floated. If the average gravity of the anthracite float at Tamaqua is assumed to be 1.60 and that of our bituminous float 1.32, then on the basis of actual volume of the solids floated, Kimberly becomes 31 pct of Tamaqua and Bessie 42 pct, which still leaves much to be desired in the way of capacity.

The test with the speed of the impellers decreased was made without supercharging. We have not used supercharging at either plant except for a short time at Bessie when that installation first went into service.

The test made by reducing air to the cell impeller was made with the air intake completely closed, but there is an annular space between the outer stand pipe and the impeller drive through which air is drawn. Closing the air intake reduced the area through which air was drawn by approximately 75 pct.

W. J. PARTON, Lehigh Navigation Coal Co. Inc., Lansford, Pa.

Laboratory Performance Tests of the Humphreys Spiral As a Cleaner of Fine Coal

by M. R. Geer, H. F. Yancey, C. L. Allyn, and R. H. Eckhouse

DISCUSSION

W. M. Bertholf—This is an excellent report of a well-conducted investigation, of sufficient scope to provide generally useful information.

Some years ago we had occasion to test the Humphreys spiral on the middling from our table plant. The conclusions reached in this investigation come as no surprise to us. In certain applications it is virtually impossible to improve on the spiral from the monetary returns point of view even though its "efficiency" may be low.

We have noted that in all successful applications of the spiral the feed is "classified," hence we were somewhat disappointed in the showing the spiral made on the Poplar Ridge coal, the only one of the four which could really be considered the equivalent of a classified feed. As was noted in the paper, however, this coal contained a considerable amount of middle and high-gravity material, apparently more than enough to make up for the theoretical advantage of small particles in these categories.

A comparison of particle size in the various specific gravity classes of the four coals is given in table XXV.

Table XXV. Comparison of Particle Size in Various Specific Gravity Classes. (Unit size, 150 m to 0.105 mm)

	Black Eagle	Clements	Poplar Ridge	Roslyn
1.3 Float	7.47	10.60	8.68	9.66
1.3 x 1.4	7.80	7.96	7.58	8.20
1.4 x 1.6	7.24	8.36	5.58	6.85
1.6 x 1.8	7.98	8.31	3.42	7.33
Sink 1.8	7.85	7.13	5.09	7.41
Avg	7.60	10.08	7.15	8.38

This tabulation indicates that there was very little difference in the size of the various gravity fractions for three of the coals, Poplar Ridge varying the most. Examination of figs. 3, 6, 7, and 8 reveals a general similarity of behavior for all coals except Black Eagle. Would the authors care to speculate on the reasons for the difference in the distribution curve for the sink 1.60 material in the washed coal of fig. 3 as compared to the others? This difference is particularly noticeable in the 8x14-mesh size. Was there a "shape effect"?

J. D. Price and **W. M. Bertholf**—The performance data on three of the coals studied appear to be typical, but there are certain departures from the general pattern in the case of the Poplar Ridge coal which are large enough to deserve special consideration. For example:

1. Despite the high proportion of sink 1.60 material in the feed, it was not possible to make a corresponding proportion of high-ash reject. (See table XVI.)

2. Even though approximately 70 pct of the feed was 1.40 float with an ash content of 6.6 pct, and this

Table XXVI. Comparison of Poplar Ridge and Roslyn Coal with Anthracite

	Ash, Pct			Total Size, Pct		
	Poplar Ridge	Roslyn	Anthracite	Poplar Ridge	Roslyn	Anthracite
8 x 14-mesh	12.1	20.1	22.9	21.6	31.3	17.9
100 x 200-mesh	22.2	24.1	36.0	5.3	7.7	10.9
Under 200	27.1	27.8	54.3	9.5	4.6	13.8

Table XXVII

Mesh	Feed to Spiral Ash, Pct	Clean Coal from Spiral Ash, Pct	Mesh	Feed to Spiral Ash, Pct	Clean Coal from Spiral Ash, Pct
3/32	14.0	10.5	60	29.7	11.3
3/64	21.2	8.3	80	30.4	14.8
28	22.6	8.5	100	30.5	22.3
35	22.9	8.5	200	43.3	33.4
48	26.0	9.9	—200	55.1	57.5

material was considerably larger than the material of higher specific gravity, the "washed coal" in the outer zones contained 16 pct ash. (See table XVII.)

3. The distribution patterns shown in fig. 7 are considerably different for washed coal and refuse from the corresponding curves for the other three coals.

All three of these "phenomena" point to some peculiarity in the nature of the high-gravity material present in this coal. Could this be particle shape?

Cubical particles would be expected to report farther out in the stream than flat particles retained on the same screen due to the increased centrifugal force and decreased drag. Could it be that the Poplar Ridge "coal" was rather slabby and the "rock" was roughly cubical? Some such feature as this, plus the large proportion of high-gravity material might account for a great deal of the abnormality of the final data.

J. F. McLaughlin and **H. H. Otto**—We wish to compliment the authors on their studies and splendid presentation of their findings. We note that two of the four coals tested were, as we view it in the anthracite area, comparatively low-ash coals and therefore not a difficult cleaning job. The other two coals, Poplar Ridge of Kentucky with 17.2 pct ash, and the Roslyn coal of Washington with 21.4 pct ash, are more like our anthracite coal cleaning problem for total ash but differ considerably by sizes as shown in table XXVI.

Our preparation plants prepare coal from various sources and numerous beds, varying somewhat in specific gravity, inherent ash and other impurities such as bone, slate, and roof rock. To successfully meet these variations and not suffer too great a loss of good material is a problem at each cleaning plant. The Hudson Coal Co. has obtained very good results on finer sizes of coal with Wilmot hydrotators and classifiers down to and including No. 5 buckwheat. The size range is No. 4 buck 3/32 x 3/64 and No. 5 buck 3/64 x 0.

The efficient classifier limit on No. 5 buck seems to be somewhere between 48 and 60-mesh. The Humphreys spirals were installed anticipating economical recoveries below this range. We find the best range for the spirals is between 3/32 and 80-mesh. Cleaning falls off at the 80 and 100-mesh, with no cleaning on —200-mesh, and usually the 150 to —200-mesh material is so high in refuse that it is uneconomical to attempt to recover it.

We agree, from our plant experience, that in the spiral as in the classifier, there is a point where particle size is very important and apparently interferes seriously with the gravity action of the cleaning equipment.

In the summary (p. 1057) the statement, "The coarsest fractions of impurities stratified so far out in the stream that it could not be removed through the refuse ports," has not proved true in our active experience as is shown in table XXVII.

Cleaning starts to fall off at 80-mesh.

"Impurities finer than 100-mesh carried out in main body of the stream and therefore were not removed in the refuse."

This is true in our coal also and, in fact, is true where coal is classified or separated by water. Currents of water cannot be adjusted to suit a range of size from 3/32 to 0. Up to a certain mesh the current is working on the difference in specific gravity of coal and refuse, then particle size takes over and floats off low and high ash particles, and, to control the total ash in the product to market requirements, the fines have to be removed.

We wish to clarify a preceding statement. The Hudson Coal Co., in practically all of its cleaning plants, has Chance process using sand as a separating medium and at Powderly the raw silt feed (3/32 x 0) is sent to the Dorr thickener and thence to the spiral plant or to the silt basin and contains fine sand. That is one of the reasons for the 80 pct ash in the refuse in this case. Powderly spiral plant prepares boiler fuel for one of our power plants, which can burn these fine sizes successfully with 16 pct to 20 pct ash. Our Loree spiral plant is for outside customers and at the moment we are doing a cleaning job with 10.5 pct to 12.0 pct of ash in the prepared coal. We have done as low as 9.9 pct ash in the coal. The refuse ash range is from 48.0 pct to 55.0 pct. We recirculate the middling product from the last turn of the spiral and we have supplemented the spiral with a launder screen 15 ft x 28 in. and clothed with 30x30-mesh screen to drop out the very fine material which is high in ash. The screen is located between the spirals and a coal tank, from which the cleaned product is pumped to settling basins. At Powderly we use a settling tank with slow-moving conveyor on the finished product which, in the overflow, carries a lot of very fine undesirable material to waste.

When only one type of coal in a confined range of size is cleaned, the equipment, whether a classifier or spiral, can be adjusted to give optimum results. This, however, is a difficult thing with us where our feed comes in part from strippings carrying clay and the bulk of it from at least four to six different seams of coal and from scattered openings.

The coal industry and equipment makers are studying the fine coal cleaning problem carefully and have made considerable advance. We have not entirely solved the problem of cleaning 80 to 100 and 200-mesh material. The final answer may be that the cost will not warrant the recovery of these sizes.

Whitman E. Brown—Everybody familiar with coal cleaning knows that whenever a device appears on the market with any merits it will only be a short time before Dr. Yancey and Mr. Geer will have some good concrete data available.

During one phase of their investigations, the authors asked us, as manufacturers, to observe the testing and offer advice for possible improvement of procedure. We were unable to offer much for improvement although we were able to show ways of obtaining more information. It is our opinion that on the samples tested the optimum results were obtained by use of the spiral alone. We all know that the response to cleaning will vary with the material. Often one machine alone cannot accomplish the full job. In the past we have been able to do much better cleaning jobs on some coals than mentioned in this paper and conversely on some fine coals we could do practically nothing.

Recently a —4-mesh coal was tested from the West Virginia, Island Creek seam. The feed was 11 pct ash raw coal and the feed rate was 20 gpm at 26.7 pct solids. The results were: (1) clean coal ash, 6.5 pct; wt, 69.1 pct. (2) refuse ash, 50.6 pct; wt, 5.7 pct. (3) middling ash, 14.7 pct; wt, 25.2 pct.

The middling was removed as a finished spiral product and fed to a sizer which produced: coal ash, 8.1 pct; wt, 79.1 pct and refuse ash, 44.0 pct; wt, 20.9 pct.

Combining the two coal and two refuse products shows: finished coal ash, 7.2 pct; wt, 89.0 pct and refuse ash, 47.3 pct; wt, 11.0 pct.

This information is submitted solely to show that the spiral-sizer combination is another effective method in attaining higher recovery at reasonable grade. Reasonable grade is a relative expression and open to discussion.

I quite agree with Dr. Yancey and Mr. Geer that a classified feed is often more desirable than a raw coal.

It was further pointed out that laboratory tests provide little or no information bearing on costs which enter into the selection of cleaning equipment. Assuming that there is a market for the product and a given grade can be maintained the next important thing is, "How much is it going to cost to obtain this product by using this equipment?"

Estimated operating costs involved in a spiral plant are as follows:

Item	Cost per Ton 2000 Lb Ton
1. Operating Labor ($1.10 per hr)	1.50
2. Power Cost, Pumping	1.30
3. Pumping Maintenance	0.35
4. Feed Distribution Maintenance	0.40
5. Product Launder Maintenance	0.04
6. Spiral Maintenance	0.50
Total Direct Spiral Plant Cost	4.09
7. Screen Maintenance	0.10
8. Dewatering Maintenance	0.10
9. Conveying Maintenance	0.10
10. Fresh Water	0.05
11. Heating (Buildings)	0.02
12. Overhead	0.10
13. Assaying and Plant Control	0.30
14. Hydraulic Sizing	0.60
15. Amortization (10 years)	2.00
Total Plant Cost per Ton	7.46

Installation cost of the equipment is approximately $1000 per ton per hour. This figure is based on cost of spirals, pumps, tanks, chutes, conveyors, piping, screens, motors, controls and wiring. By judicious planning and working under favorable conditions the installation cost may drop to $800 per ton per hour.

M. R. Geer, H. F. Yancey, and **C. L Allyn** (authors' reply)—The observation by Price and Bertholf that the Poplar Ridge coal behaved differently in the spiral than the other three coals tested is entirely correct. The explanation for the different performance on Poplar Ridge coal lies largely in the size composition of the heavy impurity. As shown in table XXVIII the tabulation of the size composition of the impurities in the four coals, the impurity heavier than 1.60 sp gr in the Poplar Ridge coal was much finer than that in the other three coals. Over 36 pct was finer than 100-mesh, and since impurity of this fineness cannot be stratified in the spiral, most of it enters the clean coal or middling products.

Table XXVIII. Screen Analyses of Impurity Heavier Than 1.60 Sp Gr in Coals Tested in Spiral, Pct

Coal	Screen Size, Mesh					
	8 to 14	14 to 28	28 to 48	48 to 100	100 to 200	Under 200
Poplar Ridge	10.2	29.6	22.3	1.7	9.3	26.9
Clements	26.8	23.1	18.7	15.6	9.4	6.4
Black Eagle	31.2	24.0	15.3	12.3	7.4	9.8
Roslyn No. 3	26.3	27.7	13.6	14.2	10.3	7.9

A second factor that caused the distribution curves for the Poplar Ridge coal to differ from those shown for the other coals is the refuse-port settings employed. Tests made on this coal with a wider refuse cut gave distribution curves which were much more similar to those for the other coals, but at a distinct sacrifice in the yield of clean coal.

Although particle shape necessarily influences the performance of the spiral, just as it modifies the effect of particle size and specific gravity in any coal-cleaning device utilizing currents of fluid, no attempt was made in this investigation to evaluate the effect of shape. It is entirely possible that, as pointed out by Messrs. Price and Bertholf, particle shape may have been responsible for some of the differences in the performance observed.

The observation by Messrs. McLaughlin and Otto that in treating anthracite in the spiral the coarsest fraction of heavy impurity does not stratify so far out in the stream that it contaminates the washed coal is interesting. The reason why anthracite should behave differently from bituminous coals in this respect is not clear, but presumably either particle shape or the relative difference in the specific gravity of the clean coal and impurity must be responsible.

J. D. PRICE and W. M. BERTHOLF, Colorado Fuel and Iron Corp., Pueblo, Colo.; H. H. OTTO and J. F. MCLAUGHLIN, Hudson Coal Co., Scranton, Pa.; and W. E. BROWN, Humphreys Investment Co., Denver, Colo.

Quantitative Efficiency of Separation of Coal Cleaning Equipment

by W. W. Anderson

DISCUSSION

John Griffen—The author has called attention to an important phase of coal cleaning since there has been considerable loose thinking regarding efficiency formulae and particularly inaccurate application of ones that have been advanced.

The unanimous acceptance of an efficiency formula has been prevented because the issue has been beclouded by the fact that the objective of cleaning—the control of chemical characteristics of the coal such as ash or sulphur content—is attained by an indirect rather than a direct method. The indirect method is usually to effect separation according to differences in specific gravity. Such separation relies on the fact that the desired chemical qualities are roughly inversely proportional to the specific gravity of the particles being separated. However, the sulphur content of the materials found in raw coal usually exhibit a far from straight line relationship with their specific gravities.

Mr. Anderson meets this issue squarely by basing his formula on the efficiency of separation according to specific gravity only. It is true that machines that effect primarily a specific gravity separation are in no way directly influenced by the chemical analysis of the materials being handled and it can be argued that it is illogical to measure their efficiency by any criterion other than that of separation according to specific gravity.

For the moment, let us accept this basis and analyze the application of the efficiency conception and formula proposed by Mr. Anderson under conditions which he has not considered but that will readily be found. Let us take the separation shown in his table III as an example. He terms this an example of a high order of quantitative efficiency. We will show the weight distribution of the various gravity fractions as between clean coal and refuse in table XV.

Only in the gravity range of 1.50 to 1.70 has material been misplaced and, on the assumption that the cleaned coal and refuse are equal in weight, Anderson shows a quantitative efficiency of 99 pct. The gravity of separation is 1.60 and one third of the 1.50 to 1.60 sp gr fraction is misplaced in the refuse while one third of the 1.60 to 1.70 sp gr fraction is misplaced in the cleaned coal.

Let us now see what effect the gravity distribution of the total products has on the quantitative efficiency calculated by his formula. Still using the data in table

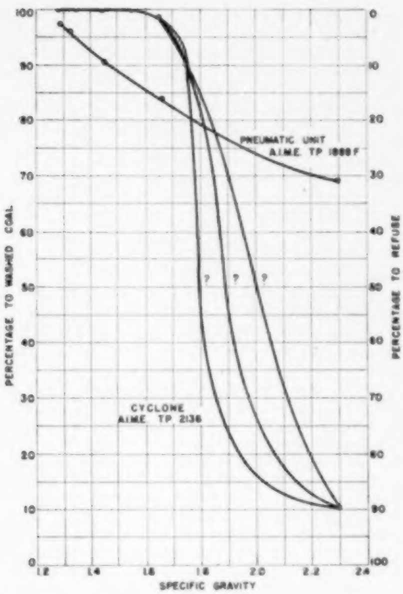


Fig. 4—Distribution curves for cyclone and pneumatic unit.

III and assuming that the clean coal and refuse are equal in weight, we need change only the total values of the 1.50 to 1.70 sp gr fractions to obtain a quite different quantitative efficiency. We need only assume that these values have doubled and immediately the amount of misplaced material is doubled and quantitative efficiency is 98 pct instead of 99 pct. The gravity of separation is still 1.60. We are still assuming that the cleaning unit is misplacing one third of the 1.50 to 1.60 sp gr material into the refuse and one third of the 1.60 to 1.70 sp gr into the cleaned coal. This is a reasonable assumption and experience shows that it is closer to the truth than the assumption that these gravity fractions would be separated in widely different proportions when their quantity is doubled.

In like manner it is readily apparent that should the total products contain one half the amount of 1.50 to 1.70 sp gr material given in table III, the quantitative efficiency would become 99.5 pct.

Such a situation immediately raises the question whether a quantitative efficiency formula that is so significantly affected by such moderate changes in the amount of 1.50 to 1.70 sp gr material is really valid and useful. That these changes are moderate is attested by the fact that they represent only a 4 : 1 change in material plus or minus 0.1 sp gr while the range of such values for all the coals of this country is perhaps 10:1. Taking only one seam of bituminous coal as an example—the Pittsburgh seam—the change from handloading to mechanical loading has often increased to 1.50 to 1.70 sp gr material from 2 pct to 6 or 8 pct.

One of the purposes of an efficiency formula is to establish figures on different coal-cleaning machines which will reveal their relative effectiveness or efficiencies in cleaning coal. As long as a formula is used which gives figures that are seriously influenced by the specific gravity distribution of the material being cleaned, no such comparisons are valid unless the materials being cleaned by two different machines are identical in their specific gravity distribution about the separating gravity.

It would appear to me that the "error area" method or some modification of it offers the most useful tool for determining the efficiency of coal-cleaning equipment which utilize primarily differences in specific gravity. As the literature shows, it is a conventional method of plotting the percent weight distribution of the various gravity fractions such as those given in the last two columns of table A.

My experience leads me to believe that a given unit of coal-cleaning equipment when fed with coals of differing specific gravity distribution will more nearly duplicate the weight distribution of identical gravity fractions than it will duplicate any other measure of separation which I know. Further, there is evidence to indicate that for a given coal-cleaning unit, the pattern of distribution of specific gravity fractions at given intervals of specific gravity each side of a separating gravity will be fairly accurately duplicated if the separating gravity is shifted by adjustments either upward or downward over the median range of gravities.

It is to be hoped that other discussers or later papers will contribute data on the "error area" method which will illuminate the points raised above. It is certain that, if these points can be established, very useful tools will be made available to study and compare the performance of coal-cleaning equipment and to more accurately predict the results that can be attained by coal-cleaning equipment when treating various coals and meeting a variety of conditions.

H. F. Yancey and **M. R. Geer—**Mr. Anderson's paper will be received with interest by all coal-preparation engineers, for the subject of washery efficiency is of unquestioned importance to everyone concerned with the cleaning of coal. Moreover, the author's contention that the basic terms "recovery," "yield," and "efficiency" should be used discerningly will meet with general approval. Even the formal discussion of AIME papers has sometimes shown the confusion that results from improper use of these terms. "Recovery" and "yield" have come to be used interchangeably to indicate the amount of cleaned product obtained in a washing operation, expressed as a percentage of the raw coal or feed. The term "efficiency," however, has no such universally accepted meaning because of the variety of efficiency formulas that have been advanced through the years.

Among the most widely used of such formulas, both here and abroad, is the one advanced by Fraser and Yancey of the Bureau of Mines in 1922, which is the ratio, expressed in percent, of the yield of washed coal actually obtained to the yield of coal of the same ash content that was available in the feed to the washing unit, the latter being determined by float-and-sink. Since this formula has been so widely used, and since the author questions its applicability, an appraisal of its usefulness in comparison with that of the substitute offered by Mr. Anderson seems desirable.

The concept of "misplaced material" is far from new, having been employed by the Bureau of Mines and by numerous European investigators for well over a decade. Those who have used it have considered it a

Table XV. Distribution of Material from Anderson's Table III

Cleaned Coal—50 pct of Combined Products
Refuse —50 pct of Combined Products

Combined Products	Specific Gravity		Pct Wt of Fraction	
	Sink	Float	Cleaned Coal	Refuse
45.0		1.30	100.0	0.0
2.0	1.30	1.40	100.0	0.0
1.5	1.40	1.50	100.0	0.0
1.5	1.50	1.60	66.7	33.3
1.5	1.60	1.70	33.3	66.7
1.5	1.70	1.80	0.0	100.0
2.0	1.80	2.00	0.0	100.0
45.0	2.00		0.0	100.0

useful criterion of washery performance but have not regarded it as an efficiency value, probably because it does not conform to an engineer's idea of efficiency as being the ratio of output to input. In other words, subtracting from 100 the percentage of raw coal that was misplaced gives a complement that might logically be termed the percentage of "properly placed" material, but although this figure indicates deviation from perfect operation, it does not constitute efficiency in the ordinary engineering sense. However, this objection to Mr. Anderson's proposed efficiency value is merely one of terminology and hence is less important than certain specific criticisms.

The principal disadvantage of the proposed formula is that it requires the use of a "distribution" or "error" curve to determine the specific gravity of separation to which the quantities of misplaced material are referred. Having pioneered in this country the use of distribution curves as a means of evaluating washery performance, we know from experience that often the number of specific gravities used in examining washery products is too few to fix the position of the curve accurately. The distribution curves for operation of a cyclone heavy-media unit shown in fig. 4 illustrate that even when as many as six densities are used in testing the washed coal and refuse, wide latitude may be possible in drawing the curve. Sometimes the data do not fix within ±0.1 the specific gravity of separation, and variances of this magnitude can change the apparent amount of misplaced material by a ratio as high as 5 : 1. In fact, if Mr. Anderson customarily tests samples of adequate weight on all of the seven specific gravities from 1.30 to 2.00 indicated in his paper he is to be commended, for few engineers feel that they can afford the cost of so much laboratory work. Hence, one of the greatest objections to the proposed formula is the inordinate cost of obtaining the data necessary to insure accurate results.

A second limitation of the proposed formula is that it simply cannot be applied to the data for some less-efficient separations, notably pneumatic units, for which the distribution curve does not cross the 50-pct ordinate even at the highest specific gravity used in testing. This condition is illustrated by the distribution curve for a pneumatic unit shown in fig. 4. With such data the specific gravity of separation, and hence the amount of misplaced material, cannot be determined.

Examples of these limitations might well have appeared in the author's paper had he used the data from actual washery operation rather than purely hypothetical figures illustrating the operation of washing units to produce washed coal of the same quality as the feed, or refuse of better quality than the washed coal. Similarly, examples assuming washed-coal yields of 99 pct or consideration of feeds containing 76 pct of material between 1.50 and 1.60 sp gr do not lend themselves readily to showing the true applicability of the formula.

The several objections to the Fraser and Yancey efficiency formula mentioned by the author, upon examination, are found to be without foundation. The first objection—that the formula gives efficiency values of over 100 pct because of the liberation of coal through degradation in the washing process—concerns a circumstance that occurs only infrequently and can be eliminated by the simple expedient of determining the amount of coal available for recovery from that present in the washery products rather than from the feed—the same procedure employed by the author.

The second objection—that the Fraser and Yancey formula is not applicable to three-product separations—is equally groundless. The yield of middling actually obtained can be related to the amount of material of that ash content present in the feed in just the same way that the efficiency of the washed-coal recovery is evaluated. The procedure is exactly the same as that employed in using the yield-ash curve to estimate the yield and quality of products obtainable from a coal by a three-product separation.

To summarize, the proportion of the feed to a washing unit that is misplaced in the wrong product is a useful criterion of washery performance, as demonstrated by its continuing use over the years. Whether this percentage should be used, or its complement employed as suggested by the author, would appear to be a matter of personal choice—one figure is as revealing as the other. Certainly, however, the author has not demonstrated that the percentage of properly placed material should be adopted as an efficiency value to replace the well-known Fraser and Yancey formula, particularly when his proposal requires a much more costly laboratory procedure to give accurate results. Both criteria are useful, and they should be considered as complementary rather than competitive.

M. G. Driessen—Whereas the efficiency of electrical or thermal engines can be clearly defined, it seems that some discussion always arises as soon as the efficiency of coal preparation equipment is described.

We would be very happy indeed if an unbiased efficiency number could be attached to each of the many coal-cleaning apparatus, but unfortunately no method acceptable to all concerned has been proposed as yet.

Mr. Anderson's paper helps us insofar as the error, or distribution curve (see fig. 1, p. 258), is mentioned which at present is the best way of describing the behavior of a wash box or coal cleaner. However the picture would only be complete if error curves would be available not only for the total feed, but also for the different size fractions. A bundle of error curves for several size fractions would give an adequate picture of the performance of a wash box.

As far as efficiency is concerned, there is a parallel between the efficiency of a thermal engine, for instance a steam turbine and a coal washer.

In operating a steam turbine it is impossible to convert the total available heat into energy. The efficiency ratio is defined by the energy actually obtained and the part of the available heat, which theoretically can be converted into energy.

In a similar way the coal operator would be interested to know which part of the saleable coal, present in the feed to the washery, would be recovered. Thus the efficiency of a cleaning apparatus should be defined as follows:

$$\text{Efficiency} = \frac{\text{Saleable coal recovered}}{\text{Saleable coal in feed}} \quad \text{or} \quad \frac{\text{Actual yield}}{\text{Theoretical yield at same ash content}}$$

This formula, at present generally accepted in Europe, had been proposed in this country by H. F. Yancey and T. Fraser as early as 1929. (Bureau of Mines *Bull.* 300). It does not seem that there is any reason to depart from this formulation of efficiency. However the formula has the following inconveniences:

The efficiency is not a constant for a certain kind of wash box, but depends on:

1. The specific gravity of separation. Expressed in other words, the efficiency depends on the definition of saleable coal or on the allowable ash content of the clean coal. It is a well-known fact, that in times of coal scarcity, almost any coal quality can be solid, and the wash box efficiency may go up to 100. Therefore a curve should be prepared for each wash box, showing the efficiency in function of the specific gravity of separation, or in function of the ash content. If the other variables (2) and (3) are kept constant, this will be the only curve necessary to judge the wash box performance.

2. The specific gravity consist and the size consist of the coal. This difficulty can be overcome by the definition of a *standardized specific gravity consist.* Whereas in a small country one standardized specific gravity consist would cover practically all the available coal seams, it might be advisable for this country

to agree on different standardized coals for different districts. Also the size consist should be standardized.

3. The efficiency depends on the capacity of the wash box. One might agree to state the efficiency at the rated capacity or else have efficiency curves for full load, overload and half load of the machine.

Conclusion

1. The best way to describe the performance of a coal-cleaning apparatus is a bundle of error curves for each individual size fraction.

2. The efficiency in which the coal operator is interested is the

$$\frac{\text{actual yield of the wash box.}}{\text{theoretical yield at the same ash content}}$$

3. These efficiencies should be plotted for a *standardized product* in function of the specific gravity of separation or in function of the desired ash content of the clean coal.

Note: By plotting the error curves on probability paper and by adjusting the abscissae with the help of logarithms, a straight line can be obtained which can be characterized by one number only. The number is called "l'ecart probable" (probable deviation) by the French and indicates the deviation from the specific gravity of separation for the ordinates at 25 or 75 pct.

W. W. Anderson (author's reply)—The comments of Messrs. Driessen, Yancey, Geer, and Griffen were greatly appreciated by the author for two principal reasons:

1. It is well known that these men have had a wide experience and have given considerable thought over a long period of time to the subject of coal cleaning equipment performance.

2. Their various discussions served to emphasize points of agreement and disagreement in regard to measures of performance.

The published literature of the AIME demonstrates how provocative is the subject of the measures which define the performance of coal-cleaning equipment; yet the fact that there is some agreement among coal preparation engineers offers hope for eventual clarification of the language and mathematics required to describe performance accurately.

As mentioned in the author's paper, numerous investigators, starting with David Hancock in 1912, have contributed fundamental ideas which have gradually expanded the basic understanding of the separations effected by coal-cleaning equipment. However, progress toward a complete understanding has been exceedingly slow, and 38 years after David Hancock's proposal of the Hancock chart there is still disagreement regarding certain vital points. Many coal preparation engineers at our coal mines today still have only the vaguest notions regarding the true performance of the equipment for which they are responsible.

Much of the recent literature in which performance values are reported deals with new types of cleaning equipment not to be found in most coal preparation plants. Moreover, the articles have been written primarily from the viewpoint of applicability, rather than from the standpoint of precise definitions of the terms used to describe performance. As a consequence, except for a few isolated instances, much of the true significance of the curves of distribution and the discussions of improperly placed material have failed to be transmitted from the academic level to the practical level of the coal-cleaning plants at the mines.

It was precisely this void which prompted the author to submit a paper dealing principally with the methods used for expressing performance. In large measure, the paper was a review of the relationship between distribution curves, gravities of separation, and the amount of improperly distributed material; and because an attempt was made to confine the discussion in a thorough manner to measures of quantitative efficiency, parts of the paper perhaps were overly simplified to the exclusion of a more comprehensive subject matter. Certainly there is much more to be said on the subject of coal-cleaning performance than is contained in the paper on quantitative efficiency; but there must be a widespread understanding of a few basic terms, such as "recovery," "yield," "efficiency," "gravities of separation," and "improper distribution," prior to the general acceptance of a more complicated subject matter dealing with bundles of error curves, error areas, and qualitative efficiencies.

Incidentally, none of the discussers mentioned qualitative efficiencies directly; but it can be inferred from some of the discussion that an acceptable method for determining qualitative efficiency, which would measure performance in terms of chemical characteristics such as ash content, would not only be a welcome additional measure of performance, but would also help to clarify some of the present confusion in respect to quantitative efficiency values.

It was particularly gratifying to learn of the unanimous approval of the discussers in regard to the difference between the basic terms "recovery" and "efficiency." These terms have been misunderstood by many coal preparation engineers, and the author attempted to demonstrate, by numerical example, the fundamental difference. Furthermore, even in recent literature there has been a misconception of the terms "efficiency" and "point of separation." Therefore, it was the hope of the author that the very simple explanations of some of these basic terms, as given in his paper, would serve to exemplify the differences in meaning. After all, if authors at the academic level are confused, it is little wonder that preparation engineers at the mines cannot discern the true significance of these terms.

There is also unanimous agreement by the discussers that curves of distribution and determination of the amount of improperly distributed material are worthwhile to the evaluation of performance. It is at this point, however, that concurrence of thought ceases.

M. G. Driessen mentions that "a bundle of error curves for several size fractions would give an adequate picture of the performance of a wash-box," and he discusses the fact that operating conditions influence the curves of error and the efficiencies of separation. John Griffen gives a numerical example to demonstrate the same point, and questions the value of an efficiency measure which is influenced by the operating conditions, specifically, in his example, the quantity of near-gravity material.

As stated in his paper, it is the opinion of the author that "the operating conditions may be the cause for the result, but they are not the result, and should not be confused therewith." The fact that efficiency varies with the operating conditions is accepted by engineers in almost all other fields of activity. Why it should be so difficult for coal-preparation engineers to accept the same basic fact with regard to coal-cleaning equipment is difficult to understand. It is not expected that a steam generating unit will have the same efficiency at half load that it does at full load; nor is it expected that an automobile engine will operate with the same efficiency at a speed of 90 mph that it will attain at 45 mph. Then why should a piece of coal-cleaning equipment be expected to have the same efficiency under one set of operating conditions that it may have under another? It simply does not make sense to anticipate any such result.

Messrs. Yancey and Geer discuss at length the value of the amount of misplaced material and admit that the efficiency determined in terms of the washery products rather than the feed is proper procedure; but they deny that the improperly distributed material can measure the efficiency of separation and doubt that the liberation of coal through degradation in the washing process concerns a circumstance that frequently occurs. An instance is cited wherein a curve of distribution on a pneumatic unit showed no point of separation, and the statement is made that the efficiency determined in terms of misplaced material "cannot be applied to the data for some less efficient separations, notably pneumatic units."

The author is in accord with Messrs. Yancey and Geer that efficiency in terms of misplaced* material cannot be determined for this particular curve because there is no point of separation in accordance with specific gravity. Hence, there can be no efficiency in terms of specific gravity.

However, this does not mean that all pneumatic operations will give a curve of this type; nor should it be inferred from the discussion of Yancey and Geer that a distribution curve such as this is peculiar to pneumatic machines alone. Many curves for pneumatic machines have crossing points at definite gravities of separation; and, hence, they can be judged for efficiency in terms of misplaced material. Moreover, curves of distribution for other types of separating equipment sometimes have no crossing points for specific sizes, indicating that no true separations in accordance with specific gravity were accomplished in these sizes.

Since the author stated in his paper that if there was no point of separation in terms of specific gravity, there could be no quantitative efficiency in accordance with his proposed formula, the example cited by Messrs. Yancey and Geer seems to be neither valid nor applicable to the argument.

Furthermore, the author also cannot concur with Yancey and Geer in their statement that "the liberation of coal through degradation in the washing process concerns a circumstance that occurs only infrequently." Rather, it is the belief of the author that this occurrence is more frequent than infrequent. The soft, low volatile coals of central Pennsylvania, southern West Virginia, and Alabama all break down in the washing process. In addition, jig plants are one of the most numerous types of cleaning devices used, and a large proportion of these jigs operate with material from secondary elevators being recirculated through a crusher to the head of the jig. In such case degradation of a portion of the feed is a premeditated, deliberate action of the operator.

In regard to the statement that the author's method requires much more costly laboratory procedure to give accurate results, this statement is definitely misleading. In the first place, the use of the Fraser and Yancey formula requires fractionation of the raw feed; whereas the author's proposed method does not. In the second place, if the test engineers have any idea at all of the approximate gravity of separation, it is not necessary to develop the complete curve of separation, but only that portion at or near the 50 pct abscissa. In the third place, the author's proposed method requires no laboratory analyses, other than float-and-sink separations; ash contents are not needed, as for the Fraser and Yancey formula.

The fact that the author's theoretical data showed test gravities from 1.30 to 2.00 should not be reason to befuddle the issue because the author purposely tried to give a complete explanation of the proposed method for the benefit of those who are not familiar with distribution curves. The proper number and choice of gravities is dependent on the judgment of the test engineers. In many cases, the gravities which determine the top and bottom ends of the curves of distribution are unnecessary; hence, they can be omitted from the test procedure. It seems obvious, however, that a theoretical discussion would not be complete without showing a full range of gravity consist.

Summarizing the discussion, there are points of agreement in regard to the basic definitions of the terms "recovery," "yield," and "misplaced material," and the proper method for determining the point of separation. However, there is still no accord in regard to an acceptable method for learning the value of the quantitative efficiency of separation.

It is strongly suspected by the author that the Fraser and Yancey formula applied to washery products rather than to the washery feed may give virtually the same answer as the author's proposed method for securing quantitative efficiency from the amount of misplaced material. However, as pointed out, the Fraser and Yancey formula requires at least the ash contents on the float-and-sink fractions, whereas efficiency determined only on weights of float-and-sink products is a quicker and cheaper method for obtaining the result.

In addition, it must be remembered that the area under the distribution curve is only another way of expressing the percent of misplaced material. Hence, it is difficult for the author to understand the need for a complete distribution curve, especially since the heavy gravity end of the curve is very expensive to attain. As most investigators know, the high gravity end of the curve is seldom a true plot of experimental data. Reference to fig. 4 in the discussion by Messrs. Yancey and Geer lends support to this statement, as will reference to published literature.

In conclusion, the author again wishes to thank Messrs. Driessen, Yancey, Geer, and Griffen for their discussions. The author also wishes to repeat that he had no intention of claiming credit for the concept of improperly distributed material, and so stated in his paper. The author does believe, however, that the worth of this concept has been underestimated. Otherwise, the proposal that it be used as a basis for defining quantitative efficiency of separation would not have been made.

J. GRIFFEN, McNally Pittsburg Mfg. Corp., Pittsburgh, Pa.; H. F. YANCEY and M. R. GEER, Bureau of Mines, Seattle, Wash.; M. G. DRIESSEN, deceased.

Operating Data for a Bird Centrifuge

by A. C. Richardson and Orville R. Lyons

DISCUSSION

F. X. Ferney—We are pleased that this paper was presented at this meeting and thank Mr. Richardson and Mr. Lyons for their effort and work in preparing it. We agree with the authors that it was unfortunate that more comprehensive data were not available when the paper was prepared, this being particularly true since we had no knowledge that the paper was being written and could not make available data which were in our process files. Further, it is our opinion that because the data are not sufficiently comprehensive the results which are presented are erratic and will lead to false conclusions. As we see it, there are three specific reasons why the data which are presented cannot be considered accurate.

1. The preparation plant was not at equilibrium while the samples were being taken. This is indicated by the fact that the percent of solids in the circulating water varied from 3.9 pct to 10.7 pct during the period when the plant was sampled.

2. We do not believe that the samples of Bird feed which were taken were representative and could indicate the true feed to the filter. Evidence of this is shown in the data where the dewatered product leaving the centrifuge contains more coarse material than the feed to it, that is, material which is +6-mesh in size.

3. We question if the data concerning screen analysis in the fine sizes is accurate and would conclude (from the data) that these tests were made by a dry-screen method. We have found that wet screening is always necessary to obtain a true picture of the quantity of —200-mesh fines which is present.

The test work described in the Lyons-Richardson paper was conducted some time ago and the tests were carried on during a period when the preparation plant had not ironed out all of their start-up problems. Since that time most of these problems have been corrected and now this is a smooth running, well-operated prep-

aration plant. Recently engineers of the Bird Machine Co. visited this plant and carried out extensive tests over a two-week period. During these tests, every effort was made to operate the plant with a closed-water system. However, it was not possible to keep the water system 100 pct closed for the quantity of fresh water entering the plant (most of this was gland water on the pumps) exceeded the quantity of water which was leaving with the coal and refuse and, therefore, a small bleed-off was necessary. This amounted to about 25 to 30 gpm. However, the bleed-off was not sufficient to prevent excessive build-up of solids in the circulating water and at the end of the test period, the solids content in the circulating water was about 25 pct. It is appreciated that this is a higher solids content than is desired for the ideal jig operation, but one of the purposes of the tests which we carried out was to determine the rate of build-up so that proper recommendations could be made to close the water system and prevent excessive solids in the circulating water. Since this preparation plant has been operating, the operators have learned that the quantity of —¼ in. x 0 coal which is entering the preparation plant is considerably greater than was first anticipated. Originally it was expected that the quantity would be between 40 and 50 tph. Actually it averages between 80 and 90 tph.

When the plant was built, just one Bird filter was installed. At times this filter handles in excess of 60 tons of coal per hour but it cannot handle all of the fines being brought into the plant. To permit acceptable plant operation, screen changes have been made, the purpose of the change being to prevent the total quantity of ¼ in. x 0 coal from passing to the sludge tank and becoming feed for the filter. The screen change which was made to replace the ¼-in. screens with 2-mm wedge wire in two thirds of the screening area. This caused a substantial quantity of the —¼-in. +10-mesh coal to remain with the coarser sizes.

The present feed to the Bird filter amounts to approximately 50 to 55 tons of coal per hour and the coal product leaving the filter carries about 14 to 15 pct moisture. The feed to the filter contains a considerable quantity of fines, averaging between 20 and 25 pct —200-mesh. The dewatered product from the centrifuge contains about 15 pct —200-mesh fines, the filtrate 7 to 8 pct solids. The solids in the filtrate are extremely fine, average particle size being about 10 to 12 microns. The operators appreciate that their overall moisture is higher than it should be and they are planning to install a second Bird filter in the near future which will allow them to handle all of their ¼ in. x 0 coal in Birds. The use of this second filter will also allow them to operate with a closed-water system and the build-up of solids in the circulating water will not exceed 15 pct. Similar results are being obtained in other plants. Of course, if it continues to be necessary to introduce more water into the plant than is removed with the product and refuse, then a small bleed-off will be required.

There are presently about 85 Bird filters operating in the coal industry. Collectively they have a capacity in excess of 50,000 tons of coal per day. When desired, it has been found possible to completely close the water circuit without excessive build-up of fines. In plants operating with a closed-water system, Bird filters are being used in two ways. If complete recovery of feed solids is desired, all of the fines can be retained in the solids product discharged from the Bird filter. Usually this will mean a ¼ in. x 0 product containing about 12 to 14 pct moisture. If a lower moisture is required, a bleed-off can be employed which can be diverted to a polishing type Bird filter. The dewatered product from the "polisher," because of its high-ash content, is usually sent to refuse and the clarified water returned to the circuit. The solids in the dewatered product from the Bird polisher usually contain from 25 to 30 pct ash.

One plant employing a system of this type with a polisher handling approximately 200 gpm has reduced the moisture in the product from the "Bird dryer" from 14 pct to 7.5 to 8 pct.

Earlier we have said that the data in the Richardson-Lyons paper indicated that accurate feed samples had not been obtained. We should point out that we have had similar experience when attempting to sample Bird feed. This is particularly true where jigs or launders are used to clean the fine coal. In plants where tables are employed, we find that the Diester distributor or similar device that splits the main stream into a number of smaller streams is an aid to sampling. Where we have been able to obtain accurate samples, we have observed that some degradation occurs in the Bird filter. The —¼-in. +6-mesh material will be reduced by approximately one third. That is, if there is 30 pct of this size material present in the feed, we will have about 20 pct retained in the cake. The —6-mesh +10-mesh will also be degraded on the order of about 10 to 20 pct. The vast majority of the coal which is broken down will report in the sizes from 14-mesh to 100-mesh. The quantity of —200-mesh is increased slightly, approximately 2 pct. That is, if there is 20 pct —200-mesh present in the feed, we find a total of about 22 pct by combining the dewatered product and the solids present in the filtrate. Most of these extreme fines, which are formed in the Bird filter, are retained in the cake and they do not have any serious effect on the circulating water.

We are hopeful that these comments together with the information obtained by Lyons and Richardson will indicate the results which are being obtained with Bird filters. It is our practice to conduct extensive tests similar to those carried out by Lyons and Richardson wherever possible.

Orville R. Lyons (authors' reply)—Relative to Mr. Ferney's objection to presentation of data obtained at a preparation plant not operating under equilibrium conditions, the authors believe that data for a plant where difficulties are being encountered or under fluctuating load conditions are of much more value to a preparation plant operator or a prospective operator than data obtained at a smoothly operating plant. The data presented show the interrelationships of some of the factors affecting the performance of one particular Bird centrifuge when the machine was being operated under a diversity of conditions.

The authors must take issue with Mr. Ferney when he questions the reliability of the samples. Every effort was made to obtain representative samples and the degree of correlation shown by the various graphs clearly proves that the samples were representative. It is true that there are some incongruities in the screen analyses presented but these screen analyses represent only a few of the samples taken and we believe that any one who has ever sampled a preparation plant will admit that it is virtually impossible to prevent the occurrence of some queer results. The authors feel that their original explanation of these discrepancies is still valid.

The authors must also disagree with Mr. Ferney's conclusion that the screen analyses were made by dry screening methods. The screen analyses were actually obtained by a combination wet and dry screening method in which the finer sizes were wet screened and only the coarsest sizes dry screened.

The authors wish to point out that they were not attempting to discredit the Bird centrifuge as a dewatering device. Their intent was and is to make available to the general public operating data concerning the Bird centrifuge. The authors appreciate Mr. Ferney's desire to point out the results to be expected when operating a Bird centrifuge under equilibrium conditions and wish that he would present a paper containing the results of the tests that the Bird Machinery Co. has conducted on its own equipment under operating conditions.

F. X. FERNEY, Bird Machine Co., South Walpole, Mass.

H — Industrial Minerals

Some Factors in Selection and Testing of Concrete Aggregates for Large Structures

by Elliot T. Rexford

DISCUSSION

G. B. Walker—In the paper reference is made to two methods of preventing reactivity between cement and aggregate in concrete structures. A third method is suggested: removing the reactive components from the aggregate before mixing. This may be done by means of a difference in the specific gravity between the desirable and the deleterious components, if a difference exists.[18] There are many commercial plants in operation making a separation between the wanted and unwanted components of mineral ores and coal, based on specific gravity difference. These plants use heavy-media separation processes in which the separation is made in a suspension of a magnetic material such as magnetite or ferrosilicon in water. The medium is held at a specific gravity between that of the desirable and undesirable components. The desirable, being heavier, will sink and the deleterious, being lighter, will float.

During 1948 and 1949 about 200,000 yd of gravel were beneficiated by heavy-media separation at the airfield of the R.C.A.F.[17] for use as concrete aggregate. At the Rivers plant, the main objective was to separate the soft shale from harder components of a local gravel deposit. This has been accomplished for two operating seasons and is a commercial and technical success. The concrete made with gravel cleaned by heavy-media separation tested slightly higher in compressive strength than the gravel from the nearest commercial source. Two seasons of operation and 200,000 yd of cleaned gravel produced have shown that, cost-wise, heavy-media separation is a process of sufficiently low cost to process economically a low-value material such as gravel.

The Rivers plant is particularly attractive from an economic standpoint because the freight rate on suitable gravel was $2.51 per yd from the nearest source.

The Rivers gravel beneficiation plant is a relatively small, prefabricated unit having a feed rate of about 80 tph. However, heavy-media separation plants having feed rates of up to 2000 tph are in operation and larger plants could be built.

The successful removal of reactive components from gravel by heavy-media separation would depend on the difference in specific gravity between the desirable and undesirable components. The specific gravity of the media used in these processes can be and is, in commercial practice, held to within ±0.01 of the desired specific gravity so that the specific gravity difference does not need to be large. In one commercial plant beneficiating magnesite, successful operation is obtained in separating materials, some of which have a specific gravity difference of as little as 0.02.[18]

Another prerequisite to successful separation by heavy-media separation is that the good and the troublesome components must be substantially free of each other at a size of about +10-mesh since the most effective size range is from 10-mesh up to 8 in., according to present commercial practice.

In certain types of concrete structures, color is of some importance. The staining and spotting of concrete by certain components in the aggregate after aging and weathering is a problem. A separation based on specific gravity difference might also prove useful in eliminating color-producing or stain-producing particles from aggregates.

Carlton Leith (in place of author)—Mr. Walker has presented an interesting commentary on the application of heavy-media separation for removal of potentially reactive constituents from concrete aggregate. Although the technique is neither new nor unique, it has not been used extensively in the processing of aggregate, partly because most operators do not wish to increase their production costs if their product is usable without special processing. In the selection of materials for large structures it is the cost per yard of the concrete in place, including all of the factors involved in placing the concrete, which will determine whether any portion of this total cost is excessive. On this basis, perhaps the prevention of reaction between cement alkalis and coarse aggregate by the elimination of potentially reactive material from a conveniently located gravel deposit is more practical than might be expected.

A more important consideration is that much of the reaction which causes deterioration of concrete results from the attack by the alkalis of the cement on constituents of the *fine* aggregate. The practical limiting grain size for effective heavy-media separation is about 10-mesh, although in cases of extreme gravity differences grains as fine as 48-mesh can be treated if dual magnetic cleaning circuits are included in the flow scheme. Thus, even if the difference in specific gravity between the potentially reactive material and the innocuous constituents of natural sand were great, which would be a very unusual condition, the lower limit of grain size for effective heavy-media separation would preclude its use for beneficiation of fine aggregate, as the average concrete mix calls for approximately 20 pct of the —No. 4 material to be finer than 48-mesh.

[16] G. B. Walker and C. F. Allen: Beneficiation of Industrial Minerals by Heavy-Media Separation. *Transactions* AIME, 184, 17: *Mining Engineering*, January 1949, TP 2503H.
[17] C. V. Trites and J. D. Shannon: Mining Process Applied to Runway Construction. *Journal* Engineering Institute of Canada (April 1949).
[18] Northwest's Magnesite's Heavy-Media Separation Plant. *Mining World* (December 1947).

G. B. WALKER, American Cyanamid Co., Stamford, Conn.

Ground Water in California

by J. F. Poland

DISCUSSION

B. C. Burgess—Prior to hearing this paper presented at the San Francisco meeting, I travelled by car from Yuma, Ariz., across south-central California and up through the San Joaquin Valley. After hearing the paper I returned by the coast route to San Diego then along the Mexican border to Yuma. The striking feature of these trips, related to this paper, was the dry river and creek beds at a time of year not the dry season. Contrasted with this was the wasteful use of water in irrigation canals and laterals.

Poland says "at least 90 pct of the ground water pumped is used for irrigation." That being the case, it would appear that first consideration should be given to conservation of this use. Probably a small part of this is actually reaching the plants. Some measure of this loss between pump and crops would be revealing.

Methods that might be employed to reduce this loss are pipe lines and spraying of some crops such as the banana planters are using in Central America. Another idea which might be investigated is to line canals and ditches with some water-repelling mineral like pyrophyllite. Perhaps a comparatively thin sprayed-on

lining of such mineral would suffice. A weed-killer might be incorporated in it.

Conservation and better utilization of the ground water supply would appear to offer a better solution of the problem than any efforts that can probably be made toward increasing recharge of the aquifers or legally restricting pumping.

J. F. Poland (author's reply)—Mr. Burgess suggests that conservation and better utilization of the ground-water supply offer a better solution than any efforts in increasing recharge of the aquifers.

In California ground-water basins, the "irrigation efficiency" (the amount of water actually consumed by plants) ranges from about 40 to possibly 90 pct. In several large areas, the efficiency has been estimated at about 50 pct, although the average for the State may be nearer 60 pct. The greater part of the water not consumed by the plants percolates to the water table and usually back into the ground-water supply, but some runs off as waste to nearby streams. Although it is inefficient to pump water and have it return to the ground-water supply, such water is not actually wasted. It is doubted that, at most, more than 10 to 20 pct of the ground water pumped for irrigation is actually wasted by running off to the sea, passing into an unusable shallow water body, or being lost through evapotranspiration by native vegetation. Thus, probably not more than one to two million acre-feet is wasted at most. Even if the greater part of this waste were eliminated by improving methods of irrigation, the total amount so conserved would be small compared to the foreseeable needs in increased recharge to ground-water basins. In the eventual full development of the San Joaquin Valley, for example, the amount of ground water required in dry periods may be several million acre-feet a year greater than present pumpage. This increased draft can be provided for only by increasing recharge and storage in wet periods.

Mr. Burgess has made a good point in stating that conservation and better utilization of the ground-water supply is needed. However, these measures will not be sufficient. The demand for ground water is increasing rapidly, and the proper development of the total water supply of the State calls for increased use of ground-water basins as storage reservoirs. Their combined capacity is far greater than that of all surface storage reservoirs that are now contemplated in California.

B. C. BURGESS, Mining Consultant, Monticello, Ga.

Industrial Mineral Economics and the Raw Materials Survey

by R. B. Ladoo and C. A. Stokes

DISCUSSION

Bruce C. Netschert—It is unfortunate that the authors of this paper consider it necessary to begin with an expression of concern over possible false interpretations of the word "economics." In their preoccupation with the definition of economics, they have adopted a definition of mineral economics which is, to say the least, unduly narrow.

Just as the broad field of economics is not confined to a study of the profitability of business concerns, but includes the problems of production, distribution, and consumption as they pertain to society as a whole; so an inclusive definition of mineral economics should not be confined to the determination of the profitability of mineral-producing and processing enterprises, but should include the significance of the unique characteristics of mineral resources as raw materials for the use of society. Such features as the exhaustibility and localized, haphazard occurrence of deposits, the existence of a secondary (scrap) supply, and the increasing cost of operation during the life of a mining enterprise are obviously factors which concern those businesses which are producing mineral raw materials, since they partially determine the profitability of such enterprises. They are also of concern to society as a whole, however, as characteristics of one of the basic elements of the economic system. In the last analysis, the contribution which mineral economics can make as a means of determining and guiding social policy with respect to the production and utilization of mineral resources is perhaps more important than its use as a basis for determining the cost accounting procedure of individual firms.

The list of "economic factors peculiar to the industrial minerals" which the authors present is in reality an application of such a broad definition of mineral economics. An inconsistency appears, however, in the inclusion of items 8 and 9 in the list. As this writer sees it, the point in question is: What influences do the characteristics of industrial minerals have on the characteristics and operating procedures of industrial mineral enterprises which are not present in the metallic mineral field? In answering this question with items 8 and 9, Messrs. Ladoo and Stokes do not recognize that there are two distinct types of differences between the two fields of enterprise. There are, on the one hand, important basic economic distinctions due to inherent economic characteristics of industrial minerals which do not pertain to metallic minerals. On the other hand are those characteristics of the industrial mineral enterprises peculiar to them alone, but which are superficial and temporary, in that they may be changed or eliminated at the discretion of the managers of those enterprises.

The lack of adequate research and development in industrial mineral production, processing, and marketing (item 8) is not due to an inherent characteristic of industrial minerals. It is true that one may perhaps describe the field of industrial mineral enterprise in terms of such a deficiency, just as one could, until recently, point to a similar lack of research and development in the coal industry; but unless it can be shown that the deficiency has been wholly or partially due to the very nature of industrial minerals themselves it is not an "economic factor peculiar to the industrial minerals" but a temporary characteristic peculiar to the industry. In the writer's opinion, the authors have not demonstrated that the former relationship exists.

Similarly, item 9, the "influence of technologic developments," is also not inherently peculiar to industrial minerals. Nowhere in the discussion of this item do the authors mention anything that is not equally applicable to the field of metallic minerals. This is not meant to imply that the specific technologic developments which the authors list are of equal significance in both fields. It does mean that such a statement as, "technological advances together with new consuming areas to provide markets make deposits commercially valuable which once were of no interest" cannot be considered as an argument that technological developments have significance in the field of industrial minerals alone.

In considering the problem of stockpiling, the authors note that the stockpiling of nonstrategic materials might be desirable if future wartime needs could exceed domestic production capacity, but dismiss this as hardly adequate to justify such stockpiling. The problem, however, can be stated in broader economic terms, i.e., the real costs (as distinguished from the money costs) of prewar *versus* wartime production. In other words, it might well be that the labor and capital required to produce a given amount of a certain mineral raw material could be used more efficiently in another industry. In such a case it is obviously advantageous to stockpile the material in prewar times rather than forego the benefits of additional production in another line of endeavor during wartime under conditions which demand the optimum use of all resources, including manpower and capital. To the writer's knowl-

edge, this issue has been largely overlooked in discussions of stockpiling policy.

The Raw Materials Survey which the authors describe is a laudable example of thinking in terms of the broad definition of mineral economics set forth above. In avoiding the policy of fostering local mineral self-sufficiency in an attempt to utilize all local mineral deposits, whatever their economic worth, the Survey, along with the economic profession, sees the goal of economic activity as the satisfaction of human wants through the utilization of the scarce means of production with the least cost to society.

BRUCE C. NETSCHERT, University of Minnesota, Duluth, Minn.

Recent Developments in the Manufacture of Lightweight Aggregates

by John E. Conley and John A. Ruppert

DISCUSSION

W. B. Mather—The growing national importance of lightweight aggregates to the construction industry makes this paper a timely presentation of the most recent developments in the field. It is presented in a clear, concise manner readily understandable by those unfamiliar with the subject of lightweight aggregates.

The classification of lightweight aggregates into two groups based on their physical properties as suggested by the authors should be adopted by the industry. The value of this dissertation however, could be enhanced by reference to a paper by Kluge, Sparks and Tuma.[25] The latter tabulates the physical properties of lightweight aggregate as compared to sand and gravel concretes.

The whole problem of expanded clay and shale aggregates has been ably presented by W. G. Bauer in a series of five papers.[26] The addition of these references to the bibliography would be of value to those interested in the subject.

The authors' discussion of the production of bloated aggregates by sintering machines is of paramount importance since many qualified individuals familiar with the industry agree that the high cost of rotary kiln operations is detrimental to the industry and that appreciable expansion of present markets can best be secured by the development of lower cost operating equipment based on sintering machines, stationary kilns, etc.

John E. Conley (authors' reply)—The authors appreciate greatly the kind remarks of Mr. Mather and concur completely with him that the subject of lightweight aggregates is timely and of great importance to the construction industry. His suggestion of bringing the literature references up-to-date is certainly an excellent idea.

References

[25] R. W. Kluge, M. M. Sparks and E. C. Tuma: Some Properties of Lightweight Concrete Aggregates. *Journal* Am. Concrete Institute (May 1949) 625-642. Also, *Pit and Quarry*, **42**, No. 3, Sept. 1949, pp. 175-177, 193.

[26] Wolf G. Bauer: Mechanics, Techniques and Economics of Expanded Clay-Shale Aggregate Production. *Pit and Quarry*, **41**, I, July 1948, pp. 71-73; II, Sept. 1948, pp. 93-96; III, Dec. 1948, pp. 91-95; IV, June 1949, pp. 87-90, and V, Nov. 1949, pp. 119-121.

Lightweight Aggregate Industry in Oregon

by N. S. Wagner and R. S. Mason

DISCUSSION

W. B. Mather—A minor recommendation that may be offered to improve the paper is the inclusion of a map of Oregon showing the general location of the various deposits. This is of especial importance to persons like myself who are not familiar with the various areas and counties mentioned.

N. S. Wagner and **R. S. Mason** (authors' reply)—We agree that an index map of Oregon showing the location of the various nonmetallic deposits would add to the paper (fig. 1).

W. B. MATHER, Southwest Research Institute, San Antonio, Texas.

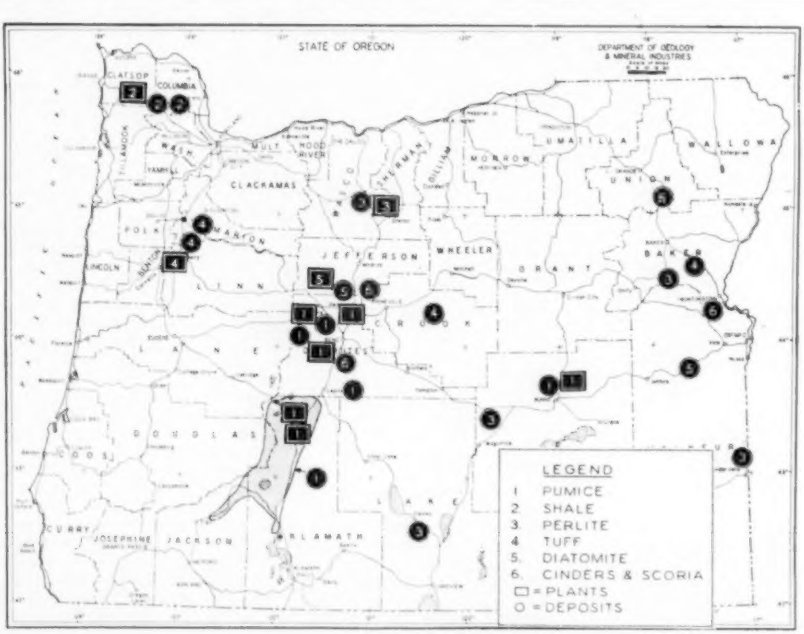


Fig. 1—Nonmetallic deposits of Oregon.

aime NEWS

Columbia Section Joins 56th Northwest Mining Association Convention in Spokane Dec. 1-2

The Columbia Section, AIME, will cooperate with the Northwest Mining Association's 56th Annual Convention in Spokane, Wash., on Dec. 1 and 2 by sponsoring three technical sessions and one luncheon. This is the Pacific Northwest's largest gathering of mining people, and the program in December will provide topics of interest in mining, milling, geology, and the problems confronting today's mining industry.

AIME President D. H. Mc Laughlin will address a noon luncheon on the convention's opening day, following general sessions that morning. That afternoon, C. A. R. Lambly, general superintendent of Pend Oreille Mines will preside over a session on drilling practices in the Northwest.

On Dec. 2, concurrent sessions on metallurgy and geology will be held, with F. H. McKinley, of Bunker Hill & Sullivan's milling department, and A. E. Weissenhorn, regional geologist for the USGS, as respective chairmen. A representative from the Department of the Interior will speak on the Defense Production Act at the noon luncheon that day, and an afternoon session will discuss the Act further. Social functions will include a dinner and dance the first evening, and the Annual Banquet on the following night.

Arizona Section to Hold One-Day Annual Meeting

The Arizona Section, AIME, will hold a one-day annual meeting at the Pioneer Hotel in Tucson on Nov. 27. The meeting will follow last year's successful pattern. AIME President D. H. McLaughlin will address the annual banquet that evening, and is expected to address the Student Society at the University of Arizona that morning. The meeting program follows:

9:30 AM—Registration.

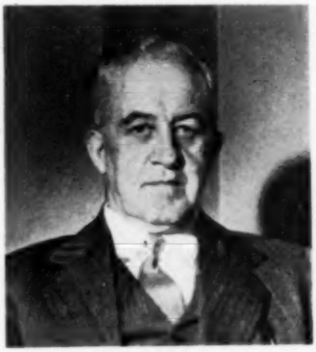
10:30 AM—Technical sessions: Underground Mining Division, Smelting Division, Ore Dressing Division.

2:30 PM — Technical sessions: Open-Pit Mining Division, Mining Geology Division.

6 PM—Cocktail party, followed by a banquet and short business meeting at which 1951 officers will be elected. Guests will be Dr. and Mrs. D. H. McLaughlin; Dr. Byron McCormick, President, University of Arizona, and Mrs. McCormick; and Roy O'Brien, Western Secretary, Mining Branch, AIME. Dancing will follow.

James B. Morrow to Receive Ramsay Medal

The Erskine Ramsay Medal for 1951 has been awarded to James Bain Morrow, vice-president of the Pittsburgh Consolidation Coal Co. This medal, which was established through a gift from Erskine Ramsay in 1948, recognizes distinguished achievement in the production, beneficiation, or utilization of bituminous or anthracite coal. The citation reads: "As an engineer and scientist, Mr. Morrow has served the coal industry with distinction. As an executive he has invariably presided with tact, judgment, and success, often through trying times of major decision. Especially notable have been his contributions to both the theory and practice of coal beneficiation, his personal leadership having been largely responsible for the present public esteem of coal preparation. He early recognized the need for coordination between the interests of the producers and consumers of coal, and has written valuable papers on this subject. In his active life he has always found time to devote to young engineers, acting as an inspiration to them, and thereby enriching the industry with young blood."


James B. Morrow

Need Geologists in Africa

The following geologic positions in the various British Colonial Geological Surveys are open. A minimum of 5 years post-bachelor experience is required. For further information, write to Chief, Alaskan and Foreign Geology, U. S. Geological Survey, Washington 25, D. C.

Nigeria: 1 senior geologist for work on mineral deposits (GS-13), 2 geologists for work on mineral deposits (GS-11 or GS-12). **Gold Coast**: 1 senior geologist for work on mineral deposits or areal mapping (GS-13), 1 geologist for work on mineral deposits or areal geology (GS-11 or GS-12). **Sierra Leone**: 1 geologist for work on mineral deposits or areal geology (GS-11 or GS-12). **Tanganyika**: 2 ground water geologists (GS-11 or GS-12). **North Borneo and Sarawak**: 1 geologist for work on fuels and paleontology (GS-11 or GS-12), 1 geologist for work on mineral deposits and petrology (GS-11 or GS-12). **Topographic Engineers**: There are a number of openings for various grades of topographic engineers.

When Changing Your Address

When notifying AIME headquarters of a change of address, or of company position or affiliation, *please mention the Branch of the Institute to which you belong*—Mining, Metals, or Petroleum. This will make for a more expeditious handling of the change and will facilitate the preparation of various reports.

Centennial of Engineering to be Held in Chicago In 1952, Marking 100th Anniversary of ASCE

Plans under way to celebrate the Centennial of Engineering in Chicago in 1952, have been announced by the Board of Directors of the American Society of Civil Engineers, meeting in Toronto.

The occasion for the celebration is the one hundredth anniversary of the American Society of Civil Engineers, founded in 1852, and the oldest of engineering societies in the United States. The entire project will provide the opportunity for American industry to place appropriate emphasis on its contributions to the advancement of civilization, and to pay tribute to the free enterprise system. In addition to the Civil Engineering group, the celebration will encompass the entire field of engineering with the other engineering societies joining forces to produce an event of international significance.

The celebration will occupy the period from approximately July 1 to September 30, 1952, and will be centered in the Museum of Science and Industry in Jackson Park, Chicago. The Museum already houses the world's greatest collection of industrial and science exhibits. Opening in early July will be a new permanent educational exhibit installation designed to bring home to visitors the tremendous contributions which engineering has made over the past one hundred years to the development of the nation and the elevation of America's standard of living. At the same time a stage production will be opened to be presented several times daily during the remainder of the summer.

Between September 3rd and 13th the greatest convocation of the engineering profession ever held in all fields and international in scope will take place. Each of the other constituent societies of the Engineers Joint Council has accepted an invitation to unite with A. S. C. E. in the Centennial Celebration. This includes the Mining and Metallurgical, Mechanical, Electrical and Chemical Engineers.

Servicemen Request Temporary Suspensions

As a result of several requests from young men currently being inducted into military service that their membership in the AIME be put on an inactive status, the following procedure was authorized by the Executive and Finance Committees.

Upon individual request from members and Student Associates entering military service in the present emergency, the Secretary is directed to offer temporary suspension of their AIME membership until such time as they may reenter professional work. No membership fees will be charged and none of the usual privileges of membership will be extended, but the member's original election date will be preserved, and he will be entitled to any credits that may be authorized on initiation fees for continuous dues-paying membership. Each such request will be handled on an individual basis, the action taken by the Secretary to be reported to the Executive Committee.

Clyde E. Weed Awarded Saunders Medal

Clyde Evarts Weed, vice-president in charge of mining operations of the Anaconda Copper Mining Co., has been selected to receive the William Lawrence Saunders Medal for 1951. It recognizes distinguished achievement in mining. It will be presented to Mr. Weed at the Annual Banquet of the AIME in St. Louis, Feb. 21, 1951.

Clyde E. Weed

The recommendation was made to the AIME Board of Directors by a committee of which William J. Coulter was chairman. The citation reads as follows:

> "Having brought to the metal mining industry a new concept of technological and operating efficiency, thus saving for the industry and for the world badly needed and valuable resources; also, for having envisioned and brought to fruition a development and exploration program, along modern and scientific lines, that is proving very helpful in the perpetuation of the industry."

AIME Board Approves 1951 Publications Committees

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Actual Operating Results Prove
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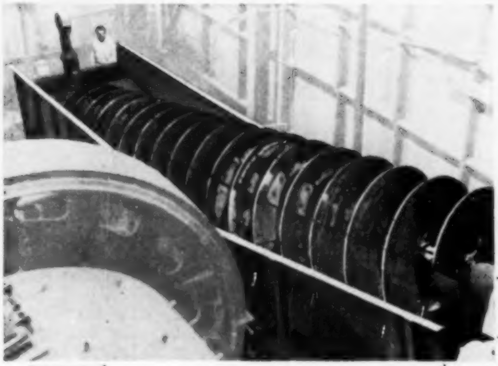

SHARP SEPARATION — From the table below note that this WEMCO S-H Classifier overflow analyzed 64.0% minus 200 mesh, showing a recovery of 88.1% of the minus 200 mesh material contained in the feed. Despite the high percentage of plus 65 mesh fraction in the classifier feed, only 4.5% was retained in the overflow. The sands, analyzing only 5.4% minus 200 mesh, contained 95.5% of the plus 65 mesh feed.

LARGE CAPACITY — Rate of overflow in this installation is 825 tons per 24 hours, exceptional capacity for a unit of this size (72" x 32' 0", Series 125, Straight Tank, Triple Pitch).

SCREEN ANALYSIS

WEMCO S-H Classifier at Fredericktown Mill, National Lead Company, St. Louis Smelting and Refining Division.

	INDIVIDUAL PERCENTAGES RETAINED ON			CUMULATIVE DISTRIBUTION OF FEED SIZES	
MESH	FEED	SAND (61.4% OF FEED)	OVERFLOW (38.6% OF FEED)	SAND	OVER-FLOW
+65	51.2%	79.8%	6.0%	95.5%	4.5%
100	9.8%	8.9%	11.2%	89.2%	10.8%
150	5.8%	3.9%	8.9%	85.0%	15.0%
200	5.0%	2.0%	9.9%	80.7%	19.3%
-200	28.2%	5.4%	64.0%	11.9%	88.1%

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Drift of Things . . . . as followed by Edward H. Robie

Provided no cataclysmic acts of God or man intervene, which is by no means unlikely, the 1950 Directory of AIME members should be mailed with this issue of the journals. Its publication has caused more grief to the Institute's secretarial staff than anything that has happened in the last two years. One of the secretaries of the other national engineering societies suggested the other day that we all get together and compare notes as to the best method of publishing a Directory, inasmuch as it seemed to be a problem for everybody. We offered to contribute a well documented memorandum on how *not* to publish one.

The 1948 Directory—the last one issued—was published in June of that year. It followed the style of previous volumes, and it cost a little more than $14,000. This year we aimed to get out a more accurate Directory at materially less cost. To make it more accurate we decided not to print it from our master file cards of members' names, positions, and addresses, since many had changed their positions without notifying us. So last March we sent a reply postal card to every member asking for his name, grade of membership, year of election, title, name of employer, address, and whether or not available for consulting work. This, we fondly thought, would give us an up-to-the-minute record of our membership, which, with a little editing, we could send to the printer. But, after waiting two months, some 5,000 of our members had not returned the card. So we had to make one out for them, from our admittedly not entirely up-to-date records.

As a further improvement in the Directory we decided to incorporate a company listing in the geographical section. That is, every company employing a member would be listed under the respective states and countries, with the names of the members, and their positions, listed in each town. This was intended to be a great convenience to members. We had the directory of the Society of Automotive Engineers as a model in this respect. It was a pious idea, with some devilish pitfalls.

As we have said, reduction of cost was one of our aims, and we thought we could eliminate about one third of the former cost. We decided to omit much of the first 115 pages of the former Directory—that part devoted to medals and awards, bylaws, the annual reports, etc., which would be printed, say, every five years. Also to be omitted were some forty pages devoted to Student Associates, since a large proportion of these addresses become obsolete at the end of a school year. A considerable further saving was expected by having the Directory set up on IBM electric typewriters and printed by offset. Another economy considered was the idea of sending a Directory only to those who request it. This is the practice followed by the AIEE for instance, and of their 37,000 members less than 5000 ask to have a Directory. This would mean a substantial saving but we felt that all AIME members had always received our Directory and would expect one without asking for it; also that even though they might not want one now they would find it useful at certain times in the future and then would not have it, necessitating more correspondence with AIME headquarters.

Some of the reasons for delay in publication and for increases over expected costs have been indicated. There were many others. We first planned to issue the book in the spring. Then by May we hoped to have it out by Sept. 1. As we write this it is Oct. 17 and we have not seen a copy yet. As the days went by, printing and binding costs increased. The ultimate cost promises to be a little more than that for the 1948 volume.

We shall be glad to have your comments for our future guidance. The book has many good points which we shall try to continue in future editions. Its faults we shall try to avoid.

Mexico City, Oct. 28-Nov. 3, 1951

Dates have now been definitely set for the AIME Regional Meeting next year in Mexico City—Oct. 28 to Nov. 3. As already announced, the meeting will be held jointly with the Instituto Nacional para la Investigacion de Recursos Minerales, and if you can't pronounce that, or don't know what it means, then it is not too early to begin to do something about it. You will get along O.K. and have a good time in Mexico City next October if you don't know a word of Spanish, but you'll have a better time if you know a bit about how they talk south of the border.

In the week before the meeting the American Mining Congress will gather at Los Angeles—Oct. 22-24—leaving three or four days to go from there to the capital of Mexico. Originally our meeting was tentatively scheduled for Oct. 15-20, but the American Society for Metals has now shifted its Detroit meeting to that week, and every effort is being made to avoid conflicts.

We are assured that the end of October is an ideal time to visit Mexico. The rainy season will have ended and the weather will be just right. Attractive trips will be arranged to such places as Taxco, Xochimilco, the Pyramids, San José de Purula, and to the old mining camp of Pachuca, still active. Even Paracutin, the new volcano, may be in eruption and worth a visit.

You can save enough money on Christmas presents alone to pay a good share of the expenses of the trip. Table silver, belt buckles, cuff links, tie pins, cigaret cases, compacts, perfume, carved wood, tooled leather, gold jewelry, blankets, serapes, glazed pottery are all good buys. Each person can bring back $200 worth, or $500 worth if they stay more than twelve days. See the September issue of Esquire as to how economical this vacation trip can be.

Just enough technical sessions to justify the trip, if you are on an expense account.

Muddled Geography

From the Baker Hotel in Dallas comes a letter which asks if we will not consider that city for a forthcoming meeting of the New England Conference of the Institute of Metals Division of the Institute. Heretofore this conference has been held in such places as Boston, Springfield, Providence, and New Haven. Doubtless Dallas offers attractions not heretofore enjoyed. How about the Petroleum Branch coming up to Boston for one of their fall meetings? Though the continental shelf along the Gulf Coast has been found to contain oil, the same shelf off New England has not yet been investigated for anything but fish.

And another convention bid: "We have just learned that your organization is planning a convention in Chicago, during the month of Chicago."

AIME Personals

Eugene S. Allen has resigned his position as mill superintendent with National Lead Co., St. Louis Smelting & Refining Div., Fredericktown, Mo. He has accepted the position of assistant superintendent of concentrators with Cerro de Pasco Copper Corp., Cerro de Pasco, Peru.

H. D. Bailey is now associated with the Molybdenum Corp. of America, Denver. He is engaged in mining and exploration work. Mr. Bailey recently resigned his position as manager of the Yellow Pine Mine at Stibnite, Idaho.

Alan T. Broderick has left the employ of the Bunker Hill & Sullivan Co., Kellogg, Idaho, and is now regional geologist with the M. A. Hanna Co. at Iron River, Mich.

Joseph F. Brown is now assistant mining engineer at the Badger mine, Anaconda Copper Mining Co., Butte.

A. F. Banfield, associate geologist of Behre Dolbear & Co., New York, has returned after spending the season directing exploration and mapping in the Yellowknife area.

Charles E. Basso has resigned his position as chief engineer with Cia. Mineras Unificada del Cerro de Potosi, Potosi, Bolivia, and is now located at 129 N. Virginia St., Reno.

R. C. Bacon has joined W. R. Grace & Co. as assistant to the vice-president. He was formerly assistant manager of Cia. Administradora de Minas, Lima, Peru.

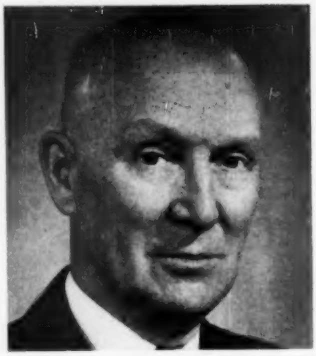
George D. Creelman, director of research, M. A. Hanna Co., has been appointed chairman of a new Bituminous Coal Research committee to anticipate technical needs and opportunities of BCR member companies and to guide the coal industry's research program accordingly. **H. J. Rose,** vice-president and director of research, BCR; **Gerald von Stroh,** director of development, BCR mining development committee, and **H. H. Lowry,** director, coal research laboratory, Carnegie Institute of Technology, are also on the committee.

R. C. Bacon

John H. Eric, geologist with the U. S. Geological Survey, was transferred from the San Francisco office to the Washington, D. C., office.

Frank M. Estes is in Indonesia making an investigation of mineral resources and a study of current economic conditions.

Lucien Eaton, consulting engineer, is on his way to Australia for H. A. Brassert & Co., New York. He had been examining iron ore concessions in northern New Quebec and on the east coast of Labrador.



Lucien Eaton

Thomas G. Fear has joined the engineering department of Clinchfield Coal Corp., Dante, Va.

E. W. Felegy, mining engineer, U. S. Bureau of Mines, Duluth, is conducting tests of radio communication and underground in mines on the Gogebic, Vermilion, Menominee and Marquette iron ranges and in the Michigan Copper country.

Edward R. Gloyd, mining engineer, U. S. Bureau of Mines, Duluth, has been recalled to active duty as a Captain in the U. S. Army.

Frederick Arthur Hames has returned to Butte to be assistant professor of metallurgical engineering at the Montana School of Mines.

Charles F. Joy is now resident geologist and engineer for the Anaconda Copper Shoshone mines division, Tecopa, Calif.

Arthur Kendall is general manager of the Central Eureka Mining Co., Sutter Creek, Calif. He was formerly superintendent of the Magnet gold mine at Geraldton, Ont.

William J. Loring, who has been president and general manager of Nevada Uranium Production Co. since its inception, has resigned. Mr. Loring will resume his practice as manager of mines and consulting engineer with headquarters at Mizpah Hotel, Tonopah, Nev.

George M. Lee has left the Macalder Mines in Kenya Colony. He is now located at 3505 W. 39th Ave., Vancouver, B. C.

E. P. Lange returned from Saudi Arabia in September. He was employed as mining engineer by the Saudi Government Bureau of Mines for the purpose of developing nonmetallic mineral deposits.

A. E. Millar of the mining department of Anaconda Copper Mining Co., recently returned from a trip to Chile. His present address is 25 Broadway, Room 1726, New York 4.

James R. Miller has accepted the position of engineer-trainee with the Island Creek Coal Co., Holden, W. Va.

Harry M. Moses has resigned as president of the H. C. Frick Coke Co. and of the U. S. Coal & Coke Co. He has been elected president of the newly organized Bituminous Coal Operators Assn., Washington.

Elmer Allen Holbrook, Dean of the Schools of Engineering and Mines, University of Pittsburgh, retired at the close of the current academic year. At that time the University presented him with a medal for distinguished service.

Elmer Allen Holbrook

John C. Fox has been appointed editor of the *Mining Congress Journal*, the monthly publication of the American Mining Congress, and is replacing **Sheldon P. Wimpfen**, who is accepting a post with the Atomic

John C. Fox

Energy Commission. Mr. Fox had been mining instructor for the past three years at the School of Mines, Columbia University, and prior to that he had a wide variety of experience in mining production and engineering work as well as working as technical representative for E. I. du Pont de Nemours & Co. He was also mining engineer for the Freeport Sulphur Co. at the Nicaro operations during the war. In the editorial field, he has been author of the "New York Letter" of the *Canadian Mining Journal* for a period of years and has made editorial contribution to the *American*

Sheldon P. Wimpfen

Metal Market. Mr. Wimpfen will be on the staff of the manager of raw materials of the AEC and will be working on special problems of uranium production. Prior to his work with the AMC, Mr. Wimpfen was assistant editor of *Mining and Metallurgy*, a Captain in the Marine Corps and had world-wide mining experience.

John T. Moran

John T. Moran has joined the Superior Coal Co. at Gillespie, Ill., as a junior mining engineer.

G. J. Matthews is now employed by the American Smelting & Refining Co., Vanadium, N. Mex., on research in mining methods.

Edward Martinez is now with Pennsylvania State College working on a research project on gob pile fires in the bituminous fields. The project is sponsored by the western and central coal operators and the State of Pennsylvania. He was formerly associated with Lafayette College.

Douglas E. Newton has been made general sales manager for the Denver Equipment Co., Denver. He was formerly plant manager for the same company.

Harry Nedwick is now employed by the Braden Copper Co., Raneagua, Chile.

George Plafker resigned his position as engineering geologist with the U. S. Army Corps of Engineers, Sacramento. He is now pursuing graduate studies in geology at the University of California.

Walter C. Stoll has joined the Benguet Consolidated Mining Co., Baguio, Philippines.

Paul V. H. Svendsen is doing metallurgical research and contact work on grinding media for Colorado Fuel & Iron Corp., Pueblo, Colo.

S. Srinivasan has been employed by Kennecott Copper Co., Bingham Canyon, Utah, on the company's student training program. He is employed as a bankman in the drilling and blasting department.

Francis A. Thomson has resigned from the presidency of Montana School of Mines due to ill health. **A. E. Adami**, school vice-president, has been appointed acting president.

Ralph A. Watson has been appointed assistant geologist, Great Northern Railway Co., Spokane, Wash. He was formerly field geologist for Anaconda Copper Mining Co., Butte.

James Westfield has been appointed chief of the accident prevention and health division of Region VIII, U. S. Bureau of Mines. He has held various positions with the Bureau for approximately 20 years.

James Westfield

Kenelm C. Winslow is planning engineer with the Warren Foundry & Pipe Co., Dover, N. J.

Shiro Yamagata is now connected with the Taihei Mining Co., Ltd., Tokyo.

—Obituaries—

Alfred H. Beebe (Member 1937), former mine manager of Cresson Consolidated Gold Mining & Metallurgy Co., has died. Born at Denver in 1891, he graduated in 1915 from the Colorado School of Mines. Following graduation he was employed as a chemist and engineer for the Portland Gold Mining Co., Victor, Colo. He served a year with the U. S. Army and following his discharge worked with several mining companies in the west. In 1929 he joined the Cresson Co. as general superintendent and remained with them until he died.

Samuel Haskell (Member 1929) died on June 27. Mr. Haskell was born in 1885 at Bay City, Mich., attended grade school at Kalamazoo and high school there and also at Albuquerque, N. Mex. He was employed as a mucker, miner, machine man and held other similar jobs, gaining experience in various mines in the western and southwestern section of the country. In 1919 he went to Colombia, S. A., and was engaged in conducting geological and engineering parties in exploring the tract of land owned by the American-Colombia Corp. Several years later he joined the Andian National Corp., Ltd., as head of the labor department and worked on the construction of oil pipe line for a distance of approximately 360 miles. In 1926 he became manager in Colombia for the Texas Petroleum Co., a subsidiary of the Texas Co. He was in charge of all field work as well as engineering on properties to be developed. In 1938 he returned to the States and was employed by the same company in their New York office.

Ernest A. Hersam (Member 1899), professor of metallurgy emeritus at the University of California, died on June 24. The professor graduated from M.I.T. in 1891 with the degree of S.B. and remained there to be an assistant in chemistry. He went to the University of California in 1892 as an analytical assistant in the mining department and in 1894 became an instructor in metallurgy. In 1923 he was appointed professor of metallurgy. He wrote numerous papers for the professional journals

on the subjects of ore dressing, ore concentration, slags, and general metallurgical interest. He also had some practical experience as a machinist. Although the professor spent most of his life on the west coast, he was born in 1868 in Stoneham, Mass.

Herman Shaw (Member 1947), former director of the Science Museum in London, has died. Dr. Shaw was born in Huddersfield, England, in 1891, attended Huddersfield Technical College and in 1914 graduated from the Imperial College of Science & Technology, South Kensington, London. Following four years as a pilot during the first World War, he joined the Science Museum as assistant director in the department of physics and geophysics. During this time he was engaged in geophysical research, and in 1935 became director of the same department. He was author of two papers published by the AIME.

Samuel A. Taylor (Member 1905), a past President of the AIME, died on August 20. Mr. Taylor was born in East McKeesport, Pa., in 1863 and was educated in public schools. He attended the Polytechnic Institute and the Western University of Pa., now the University of Pittsburgh, graduating in 1887 with the degree of civil engineer. He received the Honorary Degree of Doctor of Science from the University of Pittsburgh in 1919. In 1887 he began work for the Carnegie Steel Co. as draftsman in the new structural steel department. A short time later he resigned to accept the position of assistant engineer of construction of branch lines of the Pennsylvania Railroad until 1893 when the department was temporarily disbanded. He then began his practice as a civil and mining engineer in Pittsburgh. He had been one of the consulting engineers of the Bureau of Mines since the Bureau was established and served on several national committees pertaining to mining matters. During 1918 he assisted the Canadian government in fuel matters. From 1915 to 1920 he was a Director of the Institute, and President in 1926.

Necrology

Date Elected	Name	Date of Death
1937	Alfred H. Bebee	Unknown
1916	Thomas Ellis Brown	June 24, 1950
1918	Melville F. Coolbaugh	Sept. 9, 1950
1945	Henry A. Curtis	July 22, 1950
1915	Dudley Stuart Dean	Sept. 26, 1950
1929	Samuel Haskell	June 27, 1950
1890	Ernest A. Hersam	June 24, 1950
1926	Leo Ranney	Sept. 15, 1950
1947	Herman Shaw	Unknown
1943	John C. Southard	Sept. 4, 1950
1905	Samuel A. Taylor	Aug. 20, 1950
1911	Carl Zapffe	Aug. 28, 1950

Coming Events

Nov. 2-3, SAE, National Diesel Engine meeting, Hotel Knickerbocker, Chicago.

Nov. 3, AIME, Pittsburgh Section of Open Hearth Committee, Iron and Steel Div., and Pittsburgh Section, AIME, annual meeting, William Penn Hotel, Pittsburgh.

Nov. 3-5, New Mexico Geological Society, field conference, San Juan Basin.

Nov. 6-8, Canadian Institute of Mining & Metallurgy, annual Western meeting, Vancouver, B. C.

Nov. 9, American Mining Congress, Coal Div. Conference, William Penn Hotel, Pittsburgh.

Nov. 9-10, SAE, National Fuels & Lubricants meeting, the Mayo, Tulsa, Okla.

Nov. 14, AIME, Buffalo Section, Open Hearth Committee, Iron and Steel Div., all-day meeting, Statler Hotel, Buffalo.

Nov. 15, AIME, Western Section, Open Hearth Committee, Iron and Steel Div., Los Angeles.

Nov. 15, AIME, Mining Branch, Southern California Section, joint meeting with AAPG.

Nov. 15, AIME, Carlsbad Potash Section, Riverside Country Club, Carlsbad, N. Mex.

Nov. 16, AIME, Utah Section, Newhouse Hotel, Salt Lake City.

Nov. 16-18, Geological Society of America, annual meeting, Hotel Statler, Washington, D. C.

Nov. 17, Illinois Mining Institute, annual coal meeting and banquet, Hotel Abraham Lincoln, Springfield, Ill.

Nov. 18, AIME, Utah Section, annual fall banquet, Newhouse Hotel, Salt Lake City.

Nov. 24, AIME, Central Appalachian Section and W. Va. Coal Mining Assn., annual meeting, Greenbrier, White Sulphur Springs, W. Va.

Nov. 26-Dec. 1, American Society of Mechanical Engineers, annual meeting, Hotel Statler, New York.

Nov. 27-Dec. 2, National Exposition of Power & Mechanical Engineering, Grand Central Palace, New York.

Dec. 1-2, AIME, Columbia Section and Northwest Mining Assn., annual meeting, Davenport Hotel, Spokane.

Dec. 5, AIME, Morenci Subsection.

Dec. 7-9, AIME, Electric Furnace Steel Conference, Iron and Steel Div., Hotel William Penn, Pittsburgh.

Dec. 13, AIME, San Francisco Section, Reminiscence meeting, Engineers' Club, San Francisco.

Jan. 15-17, 1951, AIME, Minnesota Section, annual meeting. Mining symposium conducted by Center for Continuation Study, University of Minnesota.

Feb. 18-22, AIME, annual meeting, Jefferson Hotel, St. Louis. Metals Branch session to be held at the Statler Hotel.

Mar. 5-9, ASTM, spring meeting and and committee week, Cincinnati.

Apr. 2-4, 1951, AIME, Open Hearth and Blast Furnace, Coke Oven and Raw Materials Conference, Iron and Steel Div., Statler Hotel, Cleveland.

Apr. 2-5, ASME, spring meeting, Atlanta-Biltmore, Atlanta.

May 9-11, Engineering Institute of Canada, annual meeting, Mount Royal Hotel, Montreal.

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on Sept. 30, 1950, was 16,737; in addition 4184 Student Associates were enrolled.

ADMISSION COMMITTEE

E. C. Meagher, Chairman; Albert J. Phillips, Vice-Chairman; George B. Corless, H. P. Croft, Lloyd C. Gibson, Ivan A. Given, F. W. Hanson, T. D. Jones, P. Malozemoff, Richard D. Mollison, and John Sherman.

Institute members are urged to review this list as soon as the issue is received and immediately to write the Secretary's Office, night message collect, if objection is offered to the admission of any applicant. Details of the objection should follow by air mail. The Institute desires to extend its privileges to every person to whom it can be of service but does not desire to admit persons unless they are qualified.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; AM, Associate Member; S, Student Associate.

Alabama

Birmingham—Schlick, Julius R. (M).

Arizona

Safford—Beal, Laurence H. (J) (C/S—S-J).

California

Riverside—Wyman, Donald J. (J).

San Francisco—Hamilton, John W. (A).

San Francisco—Waldron, Howard L. (A) (C/S—S-M).

Winterhaven—Hardy, James G. (J) (C/S—S-J).

Colorado
Denver—Patten, Asher H. (M).
Golden—Latif, Khwaja A. (J) (C/S—S-J).
Rifle—Brown, Brinton C. (J) (C/S—S-J).

Connecticut
Westport—Cowan, Lester A. (M).

Florida
Bartow—Clawson, Floyd J. (J).
Plant City—Tillman, Chauncey G. (J)

Georgia
Cartersville—Beavers, George W. (M).

Illinois
Aurora—Johns, Burt H. (A).
Chicago—Houston, Harold B. (J) (C/S—S-J).
Morton—Jones, Albert A. (J) (C/S—S-J).
Rosiclare—Spees, Durward C. (J).

Indiana
Pendleton—Peek, Charles A. (J) (C/S—S-J).

Maryland
Baltimore—Echavez, Honorato S. (J) (C/S—S-J).

Michigan
Ann Arbor—David, Robert I. (J) (C/S—S-J).
Ishpeming—McArdle, Robert J. (J) (C/S—S-J).

Minnesota
Biwabik—Willett, James R. (J) (C/S—S-J).
Hibbing—Burton, Carlyle M. (M).
Hibbing—Mann, Richard A. (J) (C/S—S-J).
Taconite—McArthur, Charles K. (J) (C/S—S-J).

Missouri
Rolla—Arneson, Reuben H. (J) (C/S—S-J).

Montana
Butte—Carlson, Denton W. (J) (C/S—S-J).
Butte—Eastlick, John T. (J) (C/S—S-J).
Virginia City—Garrison, Rupert (M) (R. C/S—J-M).

Nevada
Reno—Hill, Starr, Jr. (M) (R/M).

New Jersey
Phillipsburg—Dye, James A. (J) (C/S—S-J).
Wharton—Gakner, Alexander (J) (C/S S-J).

New York
Bronx—Hawryluck, Alex (J) (C/S—S-J).
New York—Roberts, Lewis (J).

Ohio
Massillon—Jones, Jerome F. (A) (R. C/S—J-M).

Oklahoma
Claremore—Podpechan, Frank W. (J) (C/S—S-J).
Tulsa—Dimmick, Opie (M).
Tulsa—Vincent, Renie P. (M).

Pennsylvania
Bethlehem—Lake, Simon, III (M) (C/S—J-M).
Erie—Brauns, John W. (M).
Kingston—Corgan, John B. (M).
Kingston—Corgan, Leo R. (M).
Kingston—Corgan, William H. (M).
Kingston—Griffith, Edward M. (A).
Lebanon—Scott, James J. (J) (C/S—S-J)
Lewistown—Sheary, George W. III (J).
Lilly—Ocilka, John M. (M).
Nanticoke—Biscontini, Albert L. (M).
Nanticoke—Biscontini, Arnold L. (M).
Nanticoke—Shiffka, Frank M. (M)
Palmerton—Westphal, Warren H. (J).
Pittsburgh—Bailey, Arthur L. (M) (C/S—S-J).
Pittsburgh—Werner, James E. (J).
Plymouth—Hodgson, Ralph H. (M).

Pottersville—Martz, Lloyd L. (M).
Scranton—Fillmore, George B.
State College—Ellzey, Vernon W. (J).
Tarentum—Pollock, Calvin W. (M).
Warrior Run—Reynolds, Joseph L. (M).
Wilkes-Barre—Statkewicz, Leonard J. (M).
Windber—Greybeck, Robert J. (J) (C/S—S-J)

South Dakota
Lead—Carder, William W. Jr. (J) (C/S—S-J)
Rapid City—West, Royal F. (J) (C/S—S-J)

Texas
Big Lake—Albright, James L. (J) (C/S—S-J)
El Paso—Brown, John P. (J) (C/S—S-J)

Utah
Bingham Canyon—Clemens, Thomas F. (J) (C/S-S-J)
Keetley—Gould, Gerald E. (J) (C/S—S-J)
Salt Lake City—McElhattan, Daniel F. (M).
Salt Lake City—Parkinson, Charles J. (A).

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Spokane—Hallberg, Lyle V. (J) (C/S—S-J).
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Austria
Styria—Eichelter, Rudolf R. (M).

Canada
Alberta—Cosgrove, John C. Jr. (M) (R. C/S—S-M)
Alberta—Crouse, Bimson McL. (M).
British Columbia—Eastman, James H. (J) (C/S—S-J)
Buchans, New Foundland — Abbiss, Harry J. (J) (C/S—S-J)
Calgary, Alta—Miller, Lloyd E. (M).
Sudbury Ontario—Schneider, Vernon B. (J) (C/S—S-J)

Chile
Antofagasta—Williams, Kimball McM. (J) (C/S—S-J)
Santiago—Brown, Wendell T. (M) (R. C/S—S-M)

Dominican Republic
Ciudad Trujillo—Zoppis De Sena, Renato (M).

England
Surrey—Connor, William W. (M).

French Morocco
Casablanca—Van Dillen, Jurrien (M).

Hungary
Budapest VIII—Bolsizsar, Tibor (M).

India
Kolar Gold Fields—Ananth, Mandayam G. (M).

Japan
Shinjuku-ku, Tokyo — Shimokobe, Ryo (M).

Mexico
Chih.—Appelin, Carl W. (A) (C/S—S-A)
Mexico City—Garza Aldape, Jose M. (M) (R. M).

Peru
Casapalca—Elizondo, Joseph B. (J) (R. C/S—S-J)
Cerro de Pasco—Flow, Ralph C. (J) (C/S—S-J)

Philippines
Malate, Manila—Leitman, Samuel (M).
Quezon City—Crisostomo, Ananias T. (M) (R. C/S—S-M)

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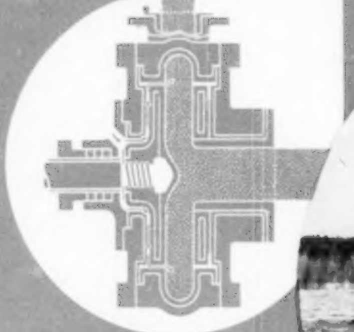
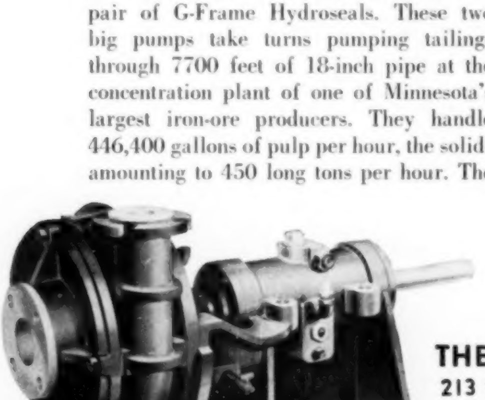
MAGAZINE ISSUED	— 23rd	" "

* Previous Issues



Discharge at tailings area



More Effective Light

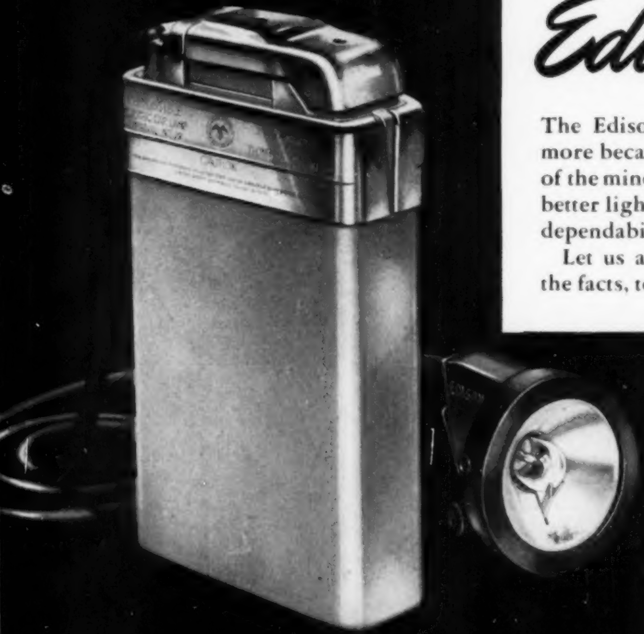
BETTER DIRECTED TO THE JOB!

Edison MODEL *R-4* ELECTRIC CAP LAMP

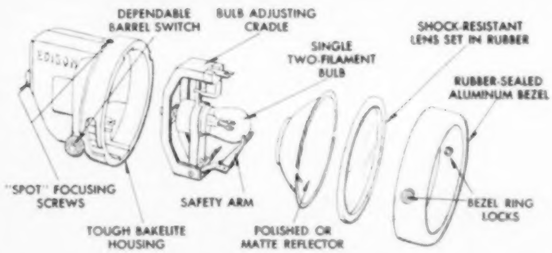
U. S. Bureau of Mines Approval No. 29

The Edison Model R-4 Electric Cap Lamp *does* more because it *has* more! This most essential tool of the miner increases tonnage with more light and better light . . . more convenience in use, complete dependability, and lower maintenance.

Let us arrange a demonstration—and give you the facts, too, on lower handling costs!



- PIERCING, LONG-RANGE SPOT OR FLOODLIGHT BEAM
- POSITIVE SPOT ADJUSTMENT
- BALANCED HEADPIECE
- SINGLE BULB WITH TWIN FILAMENTS
- STURDY CONSTRUCTION



NOTICE THE EDISON HEADPIECE REFINEMENTS

Every feature of the Edison headpiece reflects superior design engineering for better service. Headpiece is easily adapted for strong "spot" or flood beam. Simple 2-screw adjustment of spot without disassembly enables perfect focus always. The single gas-filled bulb has two filaments of equal rating, so that effective working light is always available by a turn of the barrel-switch that operates both filaments. Quick removal and re-assembly of the reflector, bulb or rubber-cushioned lens is easy as the positive bezel-locking ring requires only a 1/8 turn—no threads. Write for descriptive bulletin.

MSA SAFETY EQUIPMENT HEADQUARTERS